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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XVI.

MAY, 1887, TO FEBRUARY, 1888,

INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1888.

PHILADELPHIA:
SHERMAN & CO., PRINTERS.

PREFACE.

In consequence of the large space required in this volume for Proceedings and Papers, the list of members has been omitted. Such lists are published frequently in pamphlet form and distributed to members, and extra copies may be had at any time from the Secretary, on the terms stated in this volume, under the head of Publications.

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For the Year Ending February, 1888.

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For the term ending February, 1888:†—J. D. WEEKS, Pittsburgh, Pa.; JAMES F. LEWIS, New York City; W. B. COGSWELL, Syracuse, N. Y. *For the term ending February*, 1889:—JOHN BIRKINBINE, Philadelphia, Pa.; J. F. HOLLOWAY, New York City; E. S. MOFFAT, Scranton, Pa.

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* Prof. W. B. Potter, St. Louis, Mo., was elected President in February, 1888.

† Messrs. J. C. Bayles, New York City, A. E. Hunt, Pittsburgh, Pa., and T. Sterry Hunt, New York City, were elected Vice-Presidents in February, 1888.

‡ Messrs. Robert Forsyth, Chicago, Ill., Kenneth Robertson, Birmingham, Ala., and Charles M. Rolker, New York City, were elected Managers in February, 1888.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO MAY, 1888.

Number.	Place.	Date.	Transactions.
I.	Wilkesbarre, Pa.,*	May, 1871,	i. 3
II.	Bethlehem, Pa.,	August, 1871,	i. 10
III.	Troy, N. Y.,	November, 1871,	i. 13
IV.	Philadelphia, Pa.,	February, 1872,	i. 17
V.	New York, N. Y.,*	May, 1872,	i. 20
VI.	Pittsburgh, Pa.,	October, 1872,	i. 25
VII.	Boston, Mass.,	February, 1873,	i. 28
VIII.	Philadelphia, Pa.,*	May, 1873,	ii. 3
IX.	Easton, Pa.,	October, 1873,	ii. 7
X.	New York, N. Y.,	February, 1874,	ii. 11
XI.	St. Louis, Mo.,*	May, 1874,	iii. 3
XII.	Hazleton, Pa.,	October, 1874,	iii. 8
XIII.	New Haven, Conn.,	February, 1875,	iii. 15
XIV.	Dover, N. J.,*	May, 1875,	iv. 3
XV.	Cleveland, O.,	October, 1875,	iv. 9
XVI.	Washington, D. C.,	February, 1876,	iv. 18
XVII.	Philadelphia, Pa.,†	June, 1876,	v. 3
XVIII.	Philadelphia, Pa.,	October, 1876,	v. 19
XIX.	New York, N. Y.,	February, 1877,	v. 27
XX.	Wilkesbarre, Pa.,*	May, 1877,	vi. 3
XXI.	Amenia, N. Y.,	October, 1877,	vi. 10
XXII.	Philadelphia, Pa.,	February, 1878,	vi. 18
XXIII.	Chattanooga, Tenn.,*	May, 1878,	vii. 3
XXIV.	Lake George, N. Y.,	October, 1878,	vii. 103
XXV.	Baltimore, Md.,*	February, 1879,	vii. 217
XXVI.	Pittsburgh, Pa.,	May, 1879,	viii. 3
XXVII.	Montreal, Canada,	September, 1879,	viii. 121
XXVIII.	New York, N. Y.,*	February, 1880,	viii. 275
XXIX.	Lake Superior, Mich.,	August, 1880,	ix. 1
XXX.	Philadelphia, Pa.,*	February, 1881,	ix. 275
XXXI.	Staunton, Va.,	May, 1881,	x. 1
XXXII.	Harrisburg, Pa.,	October, 1881,	x. 119
XXXIII.	Washington, D. C.,*	February, 1882,	x. 225
XXXIV.	Denver, Col.,	August, 1882,	xi. 1
XXXV.	Boston, Mass.,*	February, 1883,	xi. 217
XXXVI.	Roanoke, Va.,	June, 1883,	xii. 3
XXXVII.	Troy, N. Y.,	October, 1883,	xii. 175
XXXVIII.	Cincinnati, O.,*	February, 1884,	xii. 447
XXXIX.	Chicago, Ill.,	May, 1884,	xiii. 1
XL.	Philadelphia, Pa.,	September, 1884,	xiii. 285
XL.	New York, N. Y.,*	February, 1885,	xiii. 585
XLII.	Chattanooga, Tenn.,	May, 1885,	xiv. 1
XLIII.	Halifax, N. S.,	September, 1885,	xiv. 307
XLIV.	Pittsburgh, Pa.,*	February, 1886,	xiv. 587
XLV.	Bethlehem, Pa.,	May, 1886,	xv. lxiii
XLVI.	St. Louis, Mo.,	October, 1886,	xv. lxx
XLVII.	Scranton, Pa.*	February, 1887,	xv. lxxvii
XLVIII.	Utah and Montana,	July, 1887,	xvi. xvii
XLIX.	Duluth, Minn.,	July, 1887,	xvi.
L.	Boston, Mass.,*	February, 1888,	xvi.
LI.	Birmingham, Ala.,	May, 1888,	xvii.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

1. The minutes of the Proceedings of each Meeting, published in pamphlet form.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. (In nearly all cases in which papers, the titles of which appear in the Proceedings, are not subsequently published, they have been withdrawn by the authors.) These papers are published separately in pamphlet form, and are marked "Subject to Revision." A small supply, beyond the edition distributed, is retained to meet subsequent demand. There are no copies on hand of papers read before 1880. The stock is nearly complete from 1880.

3. ANNUAL volumes of *Transactions*, containing the list of officers and members, rules, etc.; the Proceedings and the papers, *revised for final publication*. (In this revision after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no essential change, the correction of typographical errors and additions of later information being the usual alterations.)

4. Special editions of separate papers, for which there is demand. These are fully revised, and are usually issued in pamphlet covers.

5. Books. (Under this head the only publications thus far have been an Index to Vols. I. to X. inclusive, a Glossary of Mining and Metallurgical Terms, and a Memorial of Alexander Lyman Holley.)

All the foregoing publications are sent free to members and associates *not in arrears at the time of publication*. They are also for sale at the office of the Secretary, or are sent to purchasers by mail or express, charges paid, on receipt of the price by the Secretary, as follows:

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All communications and remittances should be addressed to R. W. Raymond, Secretary, P. O. Box 223, New York City.

RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, MAY, 1877, MAY, 1878, FEBRUARY, 1880, FEBRUARY, 1881, and FEBRUARY, 1887.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote or to be members of the Council.

Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance at the annual meeting. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, stamped with the seal of the Institute, together with a copy of this rule, and the names of the persons ineligible for election to each office. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Two other regular meetings of the Institute shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these rules from and after the publication of said announcement by the Secretary.

PROCEEDINGS.

PROCEEDINGS OF THE FORTY-EIGHTH MEETING, UTAH AND MONTANA, JULY, 1887.

LOCAL COMMITTEES.

SALT LAKE CITY, UTAH.—R. H. Terhune, *Chairman;* L. C. Karrick, C. D. Porter, C. E. Mitchener, John Sharp, Sr., P. P. Shelby, G. Lavagnino, Joseph R. Walker, R. C. Chambers, H. C. Hill, J. N. Tilemann, E. P. Ferry. L. C. Trent, C. P. Mason, W. A. Hodges, F. Dickert, Charles Read, Nich. Treweek, L. B. Rogers, J. E. Galigher, R. Mackintosh, J. C. Conklin, Frank Foote, J. Q. Packard, A. Hanauer, T. R. Jones, James Anderson, John S. Scott, O. J. Hollister, A. Faber du Faur, W. H. Bancroft. *Hotel Headquarters*—Continental Hotel.

BUTTE CITY, MONTANA.—*General Committee.—Butte City Board of Trade*—W. A. Clark, *President;* John G. Hammer, *Secretary. Reception Committee*—W. A. Clark, Marcus Daly, A. Wartenweiler, W. E. Hall, J. K. Clark, Henry Williams, C. W. Goodale, J. E. Gaylord, R. E. Booraem, J. A. Leggat, George W. Irvin II., M. J. Connell, Lee Mantle, John Caplice, H. L. Frank, M. A. Sawtelle, J. H. Harper, D. W. Tilton, P. R. Dolman, Hiram Knowles, J. C. C. Thornton. *Committee of Arrangements and Entertainment*—J. A. Leggat, C. W. Goodale, A. H. Barret, J. E. Richards, J. Kahnweiler, M. A. Sawtelle, George Haldorn, W. J. Connell, W. M. Jack, P. J. Brophy, L. R. Maillet, C. W. Ellingwood, H. L. Haupt, V. C. Whitney, John Noyes, R. B. Wallace, M. Koch. *Hotel Headquarters*—St. Nicholas Hotel.

The opening session was held in the hall of the Chamber of Commerce, Salt Lake City, on Wednesday evening, July 6. Addresses of welcome were made by Messrs. Caleb R. West, Governor of the Territory of Utah; W. S. McCormick, President of the Salt Lake Chamber of Commerce; and Richard H. Terhune, Chairman of the Local Committee, to which suitable responses were made by President Egleston and the Secretary.

The second session was held at the same place, on Friday morning, July 8th, when the following papers were read:

Gold and Silver Mining in Utah, by O. J. Hollister, Salt Lake City.

History of the Ontario Mine, Park City, Utah, by Thomas J. Almy, Salt Lake City.

The Old Telegraph Mine, Utah, by G. Lavagnino, Salt Lake City.

The third session (being the last at Salt Lake City) was held at the same place, on Friday afternoon, July 8th, when the following papers were read:

Gilsonite or Uintahite, by Joseph M. Locke, Salt Lake City (read by the Secretary in the absence of the author).

Ore- and Matte-Roasting in Utah, by R. H. Terhune, Salt Lake City.

The Sulphur Deposits of Southern Utah, by A. Faber du Faur, New York City.

The following paper was read by title:

Coal-Production in Utah, 1886, by C. A. Ashburner, Pittsburgh, Pa.

A large number of new members and associates, recommended by the Council, were unanimously elected (see list below); and after the adoption of a resolution instructing the Secretary to express the thanks of the Institute to various persons and corporations for courtesies received, the meeting was adjourned to Butte City, Montana.

A considerable number of local members, in addition to those coming from other parts of the country, attended the Salt Lake sessions, but no registry was made.

The fourth session of the meeting (being the first in Montana) was held at Butte City, Montana, on Tuesday evening, July 12th, in Caplice Hall. After an address of welcome by Mr. John A. Leggat, Chairman of the Local Committee of Arrangements, and a reply by President Egleston, the following papers were read:

The Geology of Butte, by S. F. Emmons, U. S. Geological Survey, Washington, D. C. (read by the Secretary in the absence of the author).

The Association of Minerals in the Gagnon Vein at Butte, by Richard Pearce, Denver, Col. (read by C. W. Goodale in the absence of the author).

Gold and Silver Mining and Milling at Butte, by W. P. Blake, New Haven, Conn. (read by the Secretary in the absence of the author).

The fifth session (being the second and last in Butte City) was held on Wednesday evening, July 13th, at the same place.

The following papers were read:

The Rainbow Lode, Butte City, Montana, by W. P. Blake, New Haven, Conn. (read by the Secretary in the absence of the author)

Matting Dry Auriferous Silver-Ores, by W. L. Austin, Furness, Montana.

The Chlorination of Gold-Bearing Sulphides, by E. Gybbon Spilsbury, New York City.

The Engineering Relations of the Yellowstone National Park, by Theodore B. Comstock, Champaign, Ill.

The following papers were read by title:

Further Notes on the Hydrometallurgy of Copper, by T. Sterry Hunt, Montreal, Canada.

Silver Ingot Melting at the New Orleans Mint, by F. F. Claussen, New Orleans, La.

The following persons, recommended by the Council for election as members and associates, were unanimously elected:*

MEMBERS.

Robert C. Ames,	Butte City, Montana.
Eugene Antz,	Ketchum, Idaho.
Thomas T. Baker,	Butte City, Montana.
James Baxter,	Villaldama, Mexico.
Joseph M. Bell,	Thurlow, Pa.
Robert Booraem,	Butte City, Montana.
James Breen,	Butte City, Montana.
W. L. Bretherton,	Butte City, Montana.
C. P. Brooks,	Salt Lake City, Utah.
R. H. Browne,	Salt Lake City, Utah.
Fred'k. Burbridge,	Butte City, Montana.
George H. Catlin,	Scranton, Pa.
Joseph K. Clark,	Butte City, Montana.
L. U. Colbath,	Salt Lake City, Utah.
Frank S. Craven,	Gardiner, Montana.
George R. Cullingworth,	New York City.
H. F. Dawes,	Tientsin, China.
David T. Day,	Washington, D. C.
Ferd. Dickert,	Salt Lake City, Utah.
Charles E. Dinkey,	Pittsburgh, Pa.
C. E. Dougherty,	Zaruna, Ecuador, S. A.
Francis M. Drake,	Sydney, New South Wales.
J. K. Eveleth,	Butte City, Montana.
Davenport Fisher,	Milwaukee, Wis.
John Gillie,	Butte City, Montana.
George McC. H. Good,	Houtzdale, Pa.
William E. Hall,	Butte City, Montana.
John G. Hammer,	Butte City, Montana.
Isaac B. Hammond,	Chicago, Ill.
George B. Hanna,	Charlotte, N. C.
Joseph H. Harper,	Butte City, Montana.
George J. Hillebrandt,	Butte City, Montana.
O. J. Hollister,	Salt Lake City, Utah.

* This list includes also the names of members and associates elected at the Salt Lake sessions.

L. C. Holmboe,	Chicago, Ill.
William Hoskins,	Chicago, Ill.
George W. Irvin, II.,	Butte City, Montana.
William Jenks,	Copper, New Mexico.
J. Howard Johnston,	Lima, Peru.
Lewis Jones,	Bessemer, Ala.
Adams P. King,	South St. Louis, Mo.
Oscar F. Lamm,	Lansing, Kan.
P. D. Langdon,	Chattanooga, Tenn.
William J. M. Larnach,	Wellington, New Zealand.
B. C. Leyson,	Butte City, Montana.
Detlef B. Lienau,	Bonne Terre, Mo.
Joseph M. Locke,	Salt Lake City, Utah.
William L. Loveland,	Empire, Montana.
William M. Lynch,	New Orleans, La.
G. A. McNeill,	Butte City, Montana.
Walter Mackay,	Butte City, Montana.
Charles W. Marsh,	South Bethlehem, Pa.
Auguste Mathey,	New York City.
Charles W. Matthews,	Philadelphia, Pa.
Josiah M. Merrell,	Butte City, Montana.
William G. Nebeker,	Salt Lake City, Utah.
J. G. Osborne,	Roanoke, Va.
Obed A. Palmer,	Salt Lake City, Utah.
William H. Patton,	Virginia, Nevada.
William B. Phillips,	Chapel Hill, N. C.
Charles Read,	Salt Lake City, Utah.
Francis R. Reade,	San Francisco, Cal.
Jesse W. Reno,	Boston, Mass.
F. C. Roberts,	Philadelphia, Pa.
J. C. Simpson,	St. Louis, Mo.
William H. Taylor,	Scranton, Pa.
Nicholas Treweek,	Salt Lake City, Utah.
C. B. Von Tuyl,	Rio Janeiro, Brazil.
E. A. Wall,	Salt Lake City, Utah.
R. B. Wallace,	Butte City, Montana.
Fred. M. Watson,	Zacatecas, Mexico.
Herbert A. Wilcox,	Joliet, Ill.
Henry J. Williams,	Pittsburgh, Pa.
Elliott H. Wilson,	Butte City, Montana.
Myron W. Willsie,	Rapid City, Dakota.
Alf. Woodhouse,	Transvaal, Africa.
Kinichi Yamada,	Ikuno, Japan.
Thomas York,	Butte City, Montana.
C. H. Zehnder,	Berwick, Pa.

ASSOCIATES.

Thomas J. Almy,	Salt Lake City, Utah.
Robert C. Black,	New York City.

William D. Calverley,	Crystal Falls, Mich.
P. R. Dolman,	Butte City, Montana.
E. D. Edgerton,	Helena, Montana.
W. S. Godbe,	Salt Lake City, Utah.
R. Hazard,	Peace Dale, R. I.
E. F. Holden,	Syracuse, N. Y.
John A. Leggat,	Butte City, Montana.
Othy B. Parker,	Brooklyn, N. Y.
William N. Pethick,	Tientsin, China.
S. E. Raunheim,	New York City.
Francis E. Sargent,	Butte City, Montana.
Charles H. Smyth, Jr.,	Clinton, N. Y.
Charles S. Warren,	Butte City, Montana.

After the unanimous passage of a resolution instructing the Secretary to express the thanks of the Institute for courtesies received, the session was adjourned.

The sixth and final session of the meeting was held July 17, at Mammoth Hot Springs, Wyoming, to hear a paper on the Geology of the Yellowstone National Park, by Arnold Hague, geologist in charge.

Excursions and Entertainments.

In accordance with the plan previously explained by circular from the Secretary's office, a party of members, associates and guests left Chicago, July 1, under the charge of a conductor detailed by Messrs. Thomas Cook & Son. This excursion was continued, with interruptions due to local sessions and entertainments, throughout the period of the Utah and Montana meeting, and until the arrival of the party at Duluth, July 24. It is due to the firm above named to say that the management of this journey, over a route somewhat irregular and unusual for excursion parties, was in every particular efficient and successful. All members of the party unite in hearty praise of the arrangements, so thoroughly executed, which secured to them a most comfortable and delightful trip. A day was spent at Denver, where the party received many courtesies from local members; a very interesting half-day was occupied in a rapid view of Manitou, Ute cañon, Williams' cañon, the Garden of the Gods and Glen Eyrie; and at South Pueblo, the works of the Colorado Coal and Iron Company, the Pueblo Smelting and Refining Company and the Colorado Smelting Company were visited. The party then proceeded westward by the Denver and Rio Grande railroad, passing through the Royal gorge of the Arkansas in the afternoon of July 5; over the Marshall pass at sunset of the same day; through

the Black cañon of the Gunnison that night, under a full midnight moon, and through Castle Gate and Price cañon the next morning, into the Jordan valley, where they were met, a few miles from Salt Lake city, by the representatives of the Local Committee.

Thursday morning, July 7, was spent in visits to the Germania Smelting and Refining Works and the Hanauer (formerly Morgan) Smelting Works, situated in the Jordan valley, respectively seven and six miles south of South Lake city. Transportation was furnished by special train, through the courtesy of the Utah Central Railway Company.

In the afternoon, by the courtesy of the Union Pacific Railway Company, an excursion took place to the Garfield Bathing Resort, on the shore of the Great Salt Lake, 21 miles from the city. Here the visiting members and guests enjoyed the unique luxury of a buoyant bath in the dense brine of the lake.

In the evening, a banquet was given to the Institute at the Walker House, by the citizens of Salt Lake.

On Friday afternoon, by permission of the Mormon authorities, a special organ concert was given in the Tabernacle, to permit the visiting members to hear the famous organ. At 7.30 P.M., an evening visit was made, through the courtesy of the Denver and Rio Grande Railway Company, to Lake Park Bathing Resort, the rival of Garfield, 16 miles from the city.

Saturday, July 9, was spent in a visit by special train furnished by courtesy of the Utah Central and Union Pacific railway companies, to Park City, Utah, where the Ontario, Daly and other mines and mills were examined, and the party was hospitably entertained at dinner by the Ontario Mining Company, returning in the evening to Salt Lake city.

On Monday, July 11, the travelling party left Salt Lake City, and arrived on Tuesday at Butte, Montana. It is impossible to record in detail the abundant and continuous hospitality displayed by the resident members and other citizens of Butte. Through the efficient arrangements of the Local Committee, a large number of mines and works were visited on Tuesday, Wednesday and Thursday, among which were the Gagnon, Parrott, Anaconda, Lexington, Moulton, Alice and Bluebird mines, the Parrott, Colusa, Butte and Colorado smelters, and the Lexington, Moulton, Alice and Bluebird stamp-mills. Worthy of special notice was the elegant luncheon furnished to the visiting party at the Moulton mine, by Mrs. J. K. Clark, wife of the superintendent; and a most delightful affair was

the underground collation served in handsome style to a large company of invited guests in the Bluebird mine, the workings of which, with their whitewashed timbering and incandescent electric lights, constituted an agreeable environment for this purpose.

On Thursday evening, July 14, a magnificent banquet was given to the Institute in Ransome Hall by the Butte City Board of Trade. Some 260 guests were present; music was discoursed by the Alice Band, composed of miners from the Alice mine; numerous speeches were made; and the company did not disperse until long after midnight.

Throughout the stay of the Institute at Butte, the ladies of the party were most cordially entertained by the ladies of Butte; and the gentlemen were made continual guests at the handsome rooms of the Silver Bow Club.

On Friday morning, July 15, a special train, tendered by the North Pacific Railway Company, conveyed the Institute party to Anaconda, where the old and new concentrating and reduction works of the Anaconda Company were inspected, and the party was entertained at luncheon by Superintendent Daly.

Helena, Montana, was reached about 6 P.M., and here a Local Committee pressed upon the travellers the hospitality of that city. A drive in carriages through Helena and its beautiful suburbs, and a reception at the rooms of the Montana Club, occupied all the available time; and, after this pleasant, though too brief, experience of Helena, the party proceeded on Saturday morning to Mammoth Hot Springs, in the Yellowstone National Park. Five days were spent in the Park, and on Friday, July 22, the departure from Mammoth Hot Springs terminated this episode of the journey—the climax of all its sights and wonders. Proceeding, without further stop, eastward by the Northern Pacific railroad, the excursionists arrived at Duluth, Minn., on Sunday, July 24, the day before the commencement of the Duluth meeting, having completed more than 6000 miles of travel without disaster, disagreement or delay.

PROCEEDINGS OF THE FORTY-NINTH MEETING, DULUTH, MINN., JULY, 1887.

LOCAL COMMITTEES.

DULUTH.—R. S. Munger, *Chairman;* William F. Phelps, *Secretary.*

Committee of the City Council.—Hon. J. B. Sutphin, *Mayor;* Frank W. Eaton, F. B. Daugherty, Miron Bunnell.

Committee of the Chamber of Commerce.—Messrs. R. H. Lee, C. E. Shannon, H. W. Pearson.

Committee of the Board of Trade.—Messrs. M. J. Forbes, E. A. Gilbert, C. D. Wright, George H. Dodge, J. M. Smith.

Hotel Headquarters.—Hotel St. Louis.

THE GOGEBIC REGION.—Capt. N. D. Moore, *Chairman;* A. J. Trimble, *Vice-Chairman;* C. E. Knight, *Secretary.*

Hotel Headquarters.—Chequamegon House, Ashland, Wis., and Burton House, Hurley, Wis.

The first session was held in the Opera House, Monday morning, July 25th. Mr. R. S. Munger, Chairman of the Local Committee, introduced Hon. J. B. Sutphin, Mayor of Duluth, and Mr. William F. Phelps, Secretary of the Chamber of Commerce, who made appropriate addresses of welcome, to which Mr. John Birkinbine, Vice-President, and acting as President in the absence of Prof. Egleston, replied briefly, and called on the Secretary for further remarks. After these proceedings, Mr. Birkinbine read part of a paper on the Resources of the Lake Superior Region.

The second session was held on Monday evening, in the parlor of the Duluth Boat Club. After the remainder of Mr. Birkinbine's paper had been read, the following papers were presented:

The Chapin Iron Mine, Lake Superior, by Per Larsson, Iron Mountain, Mich.

Notes on the Region North of Vermilion Lake, British Columbia, by Theodore B. Comstock, Champaign, Ill. (Read by the Secretary in the absence of the author).

The third session was held Tuesday morning, when the following papers were read:

The Canadian Iron Trade, by J. H. Bartlett, Montreal, Canada.

Two Conditions of Phosphorus in Iron—Second Paper, by Byron W. Cheever, Ann Arbor, Mich.

Experiments Illustrating the Descent of the Charge in an Iron Blast-Furnace, by Robert H. Richards and Richard W. Lodge, Boston, Mass. (Read by the Secretary in the absence of the author).

The following papers were read by title:

An Aluminum Ore, by Edward Nichols, Hermitage, Ga.

Mode of Deposition of the Iron Ores of the Menominee Range, by John Fulton, Johnstown, Pa.

Method of Mining in the Menominee Range, by John Fulton, Johnstown, Pa.

Topographical Models and their Construction, by J. H. Harden, Phœnixville, Pa. and E. B. Harden, Philadelphia, Pa.

The Incline-Railway at Lookout Mountain, Tennessee, by W. H. Adams, New York City.

Wire-Rope Haulage, and its Application to Mining, by F. C. Roberts, Philadelphia, Pa.

Silica-Determinations in Blast-Furnace Cinder, by Clemens Jones, Hokendauqua, Pa.

Inorganic Standards for the Colorimetric Carbon-Test, by Theodore W. Robinson, Joliet, Ill.

A Crystalline Sub-Sulphide of Iron and Nickel, by J. B. Mackintosh, South Bethlehem, Pa.

A New Discovery of Carbonate Iron-Ores at Enterprise, Miss., by Alfred F. Brainerd, Birmingham, Ala.

The Kaiping Coal-Mine, North China, a Report by Kwong Tung Kwang, revised and presented by J. M. Silliman, Easton, Pa.

Twenty-five Years' Progress in the Concentration of Sulphuric Acid, by W. H. Adams, New York City.

Photographing the Interior of a Coal-Mine, by F. P. Dewey, Washington, D. C.

Petroleum and Natural Gas in New York State, by Charles A. Ashburner, Pittsburgh, Pa.

The following persons, recommended as members and associates by the Council, were unanimously elected:

MEMBERS.

J. H. Adams,	Sloss Mines, Ala.
O. R. Allen,	Helena, Mont.
Clarence M. Boss,	Bessemer, Mich.
Spruille Braden,	Helena, Mont.
H. Bratnober,	Tower, Minn.
Michael Carroll,	Butte City, Mont.
Frank L. Clerc,	Cascade Locks, Oregon.
Eugene Coste,	Ottawa, Canada.
Daniel W. Cox,	Harrisburg, Pa.
Marcus Daly,	Butte City, Mont.
W. Dickerson,	Chattanooga, Tenn.

M. E. Downs,	Helena, Mont.
James Duane,	Dayton, Tenn.
Charles B. Green,	Secaucus, N. J.
J. C. Hubbell,	Crown Point, N. Y.
E. A. Lewald,	New York City.
William A. McGonagle,	Duluth, Minn.
John Mallman,	Duluth, Minn.
Elisha Morcum,	Tower, Minn.
John J. O'Farrell,	Butte City, Mont.
William J. Olcott,	Hurley, Wis.
F. C. Osgood,	Ishpeming, Mich.
James C. Pyle,	Empire, Mont.
Frank L. Sizer,	Empire, Mont.
Benjamin B. Thayer,	Butte City, Mo.
Charlemagne Tower, Jr.,	Duluth, Minn.
A. J Trimble,	Hurley, Wis.
E. M. Tutwiler,	Coalburg, Ala.
George W. West,	Birmingham, Ala.
George H. White,	Two Harbors, Mich.
T. R. Woodbridge,	Port Henry, N. Y.
Francis Wyatt,	New York City.
James E. York,	Ashland, Wis.

ASSOCIATES.

B. E. Fernow,	Washington, D. C.
R. S. Munger,	Duluth, Tenn.
George S. Potter,	New York City.

The following Associates were made Members.

W. A. Clark,	Butte City, Mont.
Frank W. Gibb,	Little Rock, Ark.
W. E. Sanders,	Helena, Mont.

A resolution was unanimously adopted, instructing the Secretary to express the thanks of the Institute by letters to the various persons and corporations, who had extended courtesies to the visiting members and guests, and the meeting was adjourned.

Excursions and Entertainments.

On Monday afternoon, July 25, an excursion was made over the St. Paul and Duluth R.R. to Thomson, 20 miles from Duluth, for the purpose of viewing the Dalles of the St. Louis River, a series of falls, aggregating nearly 500 feet.

Tuesday, July 26, was occupied with an excursion over the Duluth and Iron Range R.R. to the ore-docks at Two Harbors and the iron-ore mines at Vermilion Lake, 96 miles from Duluth.

On Wednesday afternoon, July 27, the members and guests

enjoyed a sail around the harbor of Duluth; and, in the evening of the same day, a reception at the handsome hall of the Board of Trade.

On Thursday morning, July 28, a special train conveyed the party over the Northern Pacific R.R. to Ashland, Wisconsin, where they were entertained at the Chequamegon House, and taken during the following days, as guests of the mining interests of the Gogebic range, to visit the principal mines of the range in Wisconsin and Michigan. Headquarters for these excursions were at Hurley, Wisconsin, at which point the organized and collective excursions of the Duluth meeting may be said to have ended, although many individuals and small parties continued for days to pursue in different directions their visits to the various points of interest in the Lake Superior region.

Members stopping *en route* at St. Paul, Minnesota, enjoyed the hospitality of the citizens, represented by Mr. E. P. Emerson.

Members and Associates Present.

The following members, associates and guests were registered at the headquarters of the Local Committee in Duluth. The list is probably incomplete.

A. S. Barnes.
Henry B. Barnes.
J. H. Bartlett.
John Birkinbine.
Henry W. Bulkley.
Byron W. Cheever.
F. W. Clark.
George Cullingworth.
G. M. Davidson.
B. E. Fernow.
Davenport Fisher.
E. L. Foote.
Francis T. Freeland.
R. Halnaker.
D. Garth Hearne.
Frank J. Hearne.
J. G. Hearne.
Charles S. Hinchman.
C. Russell Hinchman.
H. L. Hollis.
T. H. Hulbert.
C. Kirchhoff, Jr.
J. S. Lane.
H. M. Lane.
Per Larsson.
Alf. B. Palmer.
Richard A. Parker.
Charles O. Parsons.
George Parsons.
George S. Potter.
W. B. Potter.
A. C. Rand.
Theodore D. Rand.
Alfred Raymond.
R. W. Raymond.
C. Snelling Robinson.
D. S. Robinson.
Theodore W. Robinson.
A. M. Shook.
Pascal Shook.
J. Frank Wilkes.
Jones Wister.
Frank S. Witherbee.

PROCEEDINGS OF THE FIFTIETH (ANNUAL) MEETING, BOSTON, MASS., FEBRUARY, 1888.

LOCAL COMMITTEE.

Robert H. Richards, *Chairman*; Henry M. Howe, *Secretary*; A. S. Bigelow, F. W. Clark, T. M. Drown, W. E. C. Eustis, E. D. Leavitt, P. W. Moen, C. O. Parsons, W. P. Shinn, C. J. H. Woodbury.

Headquarters.—Hotel Brunswick.

The first session was held at the Hotel Brunswick on Tuesday evening, February 21st. Professor R. H. Richards, Chairman of the Local Committee, called the meeting to order, and introduced Gen. Francis A. Walker, President of the Massachusetts Institute of Technology, who delivered the address of welcome. President Egleston, after appropriately acknowledging, in behalf of the Institute, the cordial greeting thus extended, proceeded to deliver the Presidential Address, on the subject of Technical Education (issued separately and sent to members herewith).

The President appointed Messrs. Frank Firmstone, W. G. Neilson and J. F. Lewis as Scrutineers, to examine the ballots for officers received, and to report the result at a later session of the meeting.

The following papers were read and discussed:

Spirally-Welded Tubing, by J. C. Bayles, New York City.

The New York Mining Law, by R. W. Raymond, New York City.

The second session was held at the Massachusetts Institute of Technology on Wednesday morning, February 22d.

Mr. J. T. B. Ives, F. G. S., by invitation of the Council, read a paper on a Method of Constructing Strata-Maps, to Represent Stratification or Bedding.

The following papers were read:

Notes on the Topography and Geology of the Cerro de Pasco, Peru, by A. D. Hodges, Jr., Boston, Mass.

System of Mining in Large Bodies of Soft Ore, by Richard P. Rothwell, New York City.

The Structural Relations of Ore-Bodies, by S. F. Emmons, Washington, D. C.

During this session, Mr. Rothwell exhibited an interesting sample of coal, having a quasi-crystalline form. Mr. Jed Hotchkiss, Staunton, Va., considered it to be a case of the "cone-in-cone" structure found in some coals, and doubtless due to pressure.

The third session was held at the Massachusetts Institute of Technology, on Wednesday afternoon, February 22d, when the following papers were read:

The Electric Motor in Mining Operations, by G. W. Mansfield, Boston, Mass.

Recent Improvements in the Open-Hearth Process, by Alfred E. Hunt, Pittsburgh, Pa.

Husgafvel's Improved High Bloomary for Producing Iron and Steel Direct from the Ore, by F. Lynwood Garrison, Philadelphia, Pa.

The Bofors Steel Guns, by O. E. Michaelis, Augusta, Me.

The fourth session was held at the Massachusetts Institute of Technology, on Thursday evening, February 23d, when the following papers were read:

The Plant of the Boston Heating Company, by A. V. Abbott, New York City.

The fifth and concluding session was held at the Massachusetts Institute of Technology, on Friday evening, February 24.

The following persons recommended by the Council as members and associates, were unanimously elected:

MEMBERS.

Adachi, Jinzoo,	Pottsville, Pa.
Aertsen, Guilliaem,	Munhall Station, Pa.
Barnard, George F.,	Scranton, Pa.
Barrett, D. H.,	Baltimore, Md.
Bell, George Alfred,	Troy, N. Y.
Brown, Amos P.,	Pittsburgh, Pa.
Bowker, Charles H.,	Worcester, Mass.
Boyd, Oliver C.,	Cornwall, Pa.
Calhoun, Walter H., Jr.,	Honduras, C. A.
Carnegie, Andrew,	New York City.
Carrington, J. B,	Jasper, Ala.
Chapin, Salem T.,	Sierra Mojada, Coahuila, Mexico.
Cole, H. W. N.,	Yuscaran, Honduras, C. A.
David, W. M.,	New Brighton, S. I.
Denton, Prof. James E.,	Hoboken, N. J.
Dickerman, A. L.,	Colorado Springs, Col.
Eames, Richard, Jr.,	Providence, R. I.
Ely, Arthur C.,	Chicago, Ill.
Foote, Charles H.,	Chicago, Ill.
Gatzmer, William C.,	Philadelphia, Pa.
Gerhauser, William,	Detroit, Mich.
Glasser, C. H.,	North Tarrytown, N. Y.

Goodrich, Robert R.,	Stone Cliff, W. Va.
Green, George R.,	East Orange, N. J.
Hean, B. F.,	Cornwall, Pa.
Heidenreich, E. L.,	Chicago, Ill.
Hewitt, Robert A.,	Anniston, Ala.
Hillard, Charles J.,	Penna. Furnace, Pa.
Hodge. James M.,	Big Stone Gap, Va.
Hooper, Edward,	Garfield, Nev.
Humbert, George J.,	Pottstown, Pa.
Hunt, William P., Jr.,	Boston, Mass.
Inglis, Alexander,	Cloverport, Ky.
Jenney, Royal A.,	Newberry, Mich.
Jones, James F.,	Philadelphia, Pa.
Kada, Teichi,	Japan.
Kedzie, George Edward,	Ouray, Col.
Kingsbury, Joseph T.,	Salt Lake City, Utah.
Knorr, Frederick H.,	South Bethlehem, Pa.
Kreissmann, Charles,	Port Arthur, Ontario, Canada.
Kwang, Kwong Yung,	Kaiping Coal Mine, North China.
Littell, Walter A.,	Candelaria, Nev.
Locke, B. H.,	Central City, Col.
Louis, Henry,	London, England.
Lyle, D. A.,	Boston, Mass.
Mansfield, George W.,	Melrose Highlands, Mass.
McLure, Charles D.,	St. Louis, Mo.
Mack, Thomas A.,	Oxmoor, Ala.
Marsh, Walter,	New South Wales.
Means, Ellison C.,	Low Moor, Va.
Milner, H. Key,	Birmingham, Ala.
Newell, Fred. H.,	Boston, Mass.
Norton, L. H.,	Bridgeport, Conn.
Nostrand, P. Elbert,	Brooklyn, N. Y.
Ord, William Davidson,	Jeddo, Pa.
Otten, Arthur H.,	Ironton, Ohio.
Patterson, S. B.,	Dover, N. J.
Patterson, William E.,	Burke City, Idaho.
Pettibone, Aug.,	Bodie, Cal.
Phipps, Henry,	Pittsburgh, Pa.
Pistor, William,	New York City.
Rich, William,	Minas de Rio Tinto, Spain.
Richmond, Fred. C.,	Shullsburg, Wis.
Schlesinger, William M.,	Philadelphia, Pa.
Seddon, Thomas,	Birmingham, Ala.
Setterwall, H. A. F.,	Worcester, Mass.
Shields, Charles O.,	Sierra Mojada, Coahuila, Mexico.
Sprague, T. W.,	Boston, Mass.
Stearns, Geo. M.,	Boston, Mass.
Straker, Tooke,	Boonton, N. J.
Surls, Joseph K.,	Anniston, Ala.
Sykes, John H.,	Worcester, Mass.
Taft, Harry H.,	Ishpeming, Mich.

Talbot, Henry P.,	Boston, Mass.
Trotz, J. O. Emanuel,	Worcester, Mass.
Van Wickle, Augustus S.,	New York City.
Waithman, Hubert,	Pinos Altos, Chihuahua, Mexico.
West, Alfred R.,	Denver, Colo.
Wetmore, Ernest C.,	Detroit, Mich.
Williams, Frank,	South Bethlehem, Pa.
Williams, H. D.,	Bessemer, Mich.
Williams, John,	New York City.
Wood, Alan, Jr.,	Conshohocken, Pa.
Wood, Howard,	Conshohocken, Pa.

ASSOCIATES.

Bigelow, Albert S.,	Boston, Mass.
Coxe, Alexander B., Jr.,	Drifton, Pa.
Escobar, Francisco,	New York City.
Gifford, Stanley D.,	Tuckahoe, N. Y.
Gaines, A. P.,	Boston, Mass.
Mahony, Arthur S. J.,	New York City.
Seddon, Richard John, M. H. R.,	Kumara, Westland, N. Z.

The following persons, upon recommendation of the Council, were changed from associates to members: W. M. Claypool, J. H. Janeway, Jr., Theodore Stevens, E. S. Stackhouse, H. A. Tuttle, Walter Wyckoff.

Upon the recommendation of the Council, the following persons were elected honorary members of the Institute:

Daubrée, A.,	Paris, France.
Goupillière, Haton de la,	Paris, France.
Hoefer, Hanns,	Leoben, Austria.
Posepny, Franz,	Przibram, Bohemia.

The Scrutineers appointed at the first session reported the following officers as elected:

PRESIDENT.

William B. Potter,	St. Louis, Mo.

VICE-PRESIDENTS.

(To serve two years.)

J. C. Bayles,	New York City.
A. E. Hunt,	Pittsburgh, Pa.
T. Sterry Hunt,	Montreal, Can.

MANAGERS.

(To serve three years.)

Robert Forsyth,	Chicago, Ill.
Kenneth Robertson,	Birmingham, Ala.
C. M. Rolker,	New York City.

TREASURER.

THEODORE D. RAND, ... Philadelphia.

SECRETARY.

ROSSITER W. RAYMOND, ... New York City.

The Secretary read the

ANNUAL REPORT OF THE COUNCIL.

In accordance with the rules, the Council makes the following report to the Institute:

The financial statements of the Secretary and the Treasurer, duly audited, show receipts from all sources for the year ending January 31st, 1888, including the balance at the beginning of the year, of $23,282.46, and expenditures of $18,667.31, including the amount expended in the purchase of United States bonds, leaving a surplus of $4,615.15. The Treasurer now holds United States bonds belonging to the Institute, as follows: 4 per cent. bond, par value $2900; 4½ per cent. bonds, par value, $4400; total, $7300. The present market value is about $8376. Excluding from the statement both the balance from last year and the amount invested, the actual receipts of the year were $2829.20 greater than the expenses. This increase is due partly to the larger number of members, and partly to the larger sale of publications, principally back-volumes of the *Transactions*.

The detailed statements of the Secretary and the Treasurer are consolidated in the following classified statement:

Statement of the Secretary and Treasurer, of Receipts and Disbursements, from February 1st, 1887, to January 31st, 1888.

Received from last statement,		$ 4,379 45	
Received for dues from members and associates,		15,032 94	
" " life-memberships,		1,200 00	
" " associateships,		100 00	
" " sale of publications,		1,112 22	
" " binding Transactions,		756 36	
" " authors' pamphlets,		80 38	
" " engraving and electrotypes,		115 25	
" " interest on United States bonds,		505 86	
			$23,282 46

Brought over,			$23,282 46
Paid for printing volume xv. Transactions,	2,244 73		
" " " pamphlet editions of papers,	1,249 05		
" " " authors' editions of papers,	54 35		
" " " list of members,	139 50		
" " " mailing list,	38 75		
" " " circulars, ballots, etc.,	130 60		
" " binding volume xv. Transactions,	860 21		
" " " other volumes of Transactions,	115 75		
" " " miscellaneous exchanges,	108 15		
" " engraving and electrotyping,	1,197 00		
" " postage (including P.O. Box rent),	921 87		
" " incidental expenses of mailing books,	39 54		
" " stationery (including postpaid envelopes),	321 77		
" " rent of offices,	800 00		
" " janitor's fees,	114 00		
" " express charges, freight, etc.,	113 94		
" " telegrams,	16 92		
" " index of volume xv. of Transactions,	100 00		
" " " " " i.–xv. inclusive,	1,075 71		
" " storage of Transactions (adv. for 1888),	151 01		
" " insurance,	5 00		
" " coal purchased,	25 00		
" " rent of safe deposit box,	5 00		
" " fitting up additional room,	418 23		
" " small expenditures (unclassified),	59 59		
" " salaries of secretary, assistants and stenographer,	4,740 00		
" " expenses of secretary at meetings,	929 14		
" " type-writing machine,	100 00		
" " U. S. Government bonds purchased,	2,592 50		
			$18,667 31
Balance,			4,615 15
			$23,282 46

The expenses of the year include the cost of preparing, but not that of printing the elaborate Index volume, now in press, and containing a complete analytical index of Vols. i.–xv. inclusive, of the *Transactions*. It has been found impossible to issue this volume, as intended, before the present meeting; but it will appear as soon as the necessary careful proof-reading will permit, and the Council has decided to send it free not only to all members of 1887 not in arrears, but also to members elected in 1888.

Three meetings have been held during the year: the annual meeting at Scranton, Pa., in February; a meeting in Utah and Montana, with sessions at Salt Lake City and Butte City respectively, during July; and a meeting at Duluth, Minn., at the end of the same month. These meetings were accompanied with many interesting

and delightful excursions, and resulted in large additions to the membership as well as many valuable contributions to the *Transactions* of the Institute.

Changes in membership since the last annual report have been as follows: 200 members and 29 associates have been elected, and have accepted their elections; 7 members have been reinstated; 23 members and 12 associates have resigned; 56 members and 11 associates have been dropped from the roll (subject to reinstatement at the discretion of the Council) for non-payment of dues; and the deaths of 11 members and 1 associate have been reported.

The list of deaths comprises the names of William Brückner, Edward Gridley, A. Harnickell, Jackson Bailey, Robert Bunsen, Isaac E. James, B. F. Emerson, James B. White, David B. Lewis, Charles F. Manness, and Frederick Mercur, members; and J. R. D. Owen, associate.

These changes are tabulated as follows:

	H. M.	F. M.	M.	A.	Total.
At date of last report	7	48	1354	176	1585
Gain by Election			200	29	229
Change of Status			9		9
Reinstatement			7		7
Losses by Resignation			23	12	35
Dropping			56	4	60
Change of Status		1		8	9
Death			11	1	12
Total Gains					245
Total Losses					116
Present Membership	7	47	1480	180	1714

The apparently large number of members dropped for non-payment of dues includes many who will doubtless be subsequently reinstated, and many others whose present addresses are not known and who may have died. In any event, it is better that the names of such as are not heard from should be removed from the list of active members. The Institute is large and strong enough not to care for the mere show of numbers. The general rules governing the treatment of members in arrears are as follows (with discretionary modifications in special cases):

1. Members and associates are desired and expected to pay their dues promptly on receipt of the first notice, which is sent immediately after the annual meeting.

2. The volumes of *Transactions* and the special extra publications of the Institute are not sent to persons in arrears for the year during which they are published.

3. The papers, in provisional pamphlet edition, and all circulars from the Secretary are sent to persons in arrears until the amount of arrears is at least twenty dollars, except that, upon the return of mail-matter through the Post-office, or other evidence showing that the address has been changed and is unknown to the Secretary, the sending of all publications is immediately discontinued.

4. Members and associates in arrears are notified at least three times in each year and requested to pay the same. Efforts are also made by correspondence to ascertain addresses which have not been reported to the Secretary, and all possible means are taken to avoid the dropping of names without ample notice actually received by the persons concerned.

5. Resignations are accepted at any time from members or associates not in arrears, or from those in arrears when accompanied by payment of the same. In administering this rule a liberal construction is put upon it. Thus, although the dues are strictly payable in advance "at the annual meeting," it is customary not to expect payment until notice has been served, after that meeting. On receipt of such first notice, any member or associate, though technically in arrears, may resign without paying the dues of that year; and this privilege is usually extended to the period of the May meeting. After that meeting, and after further notice or notices given, it is usually held that a member or associate resigning should accompany his resignation with the payment of dues for the year.

6. In cases of present inability to pay the dues, of which notice is promptly given to the Secretary, accompanied by assurances of the desire to retain membership and the intention to discharge the debt, every reasonable accommodation is cheerfully extended. In the vast majority of cases, the names finally dropped are those of persons who make no reply to letters and circulars on the subject.

7. While members or associates dropped for non-payment may be reinstated by the Council upon future full payment of arrears, such reinstatement cannot be claimed as a right; and application for it should be accompanied with explanation of the circumstances attending the default.

8. On the other hand, persons once dropped for non-payment of dues and wishing to re-enter the Institute without payment of all past arrears to the date of their new application, may be proposed

and re-elected as new members after an interval of not less than three years, counting from the date when they were dropped. But in that case the Council, before recommending such candidates for re-election, will consider the existence of the unpaid debt as an objection requiring removal by satisfactory explanation or payment. As a general rule, payment will be expected. Thus, for instance, a member dropped when in arrears for two years, desiring three years later to be reinstated, may be restored without re-election upon payment of five years' dues, or he may be re-elected upon the recommendation of the Council, which would probably be given on payment of the two years' dues unpaid at the time of suspension.

These explanations are given for the information of the increasing number of applicants for reinstatement or re-election.

Notice was given by the Council of the following amendments to the rules, to be acted upon at the next annual meeting:

It is proposed to amend Rule II. by striking out the final proviso, and Rule V. by striking out the parenthesis, "(except honorary members)."

The following papers were read by title:

The La Plata del Libano Mines, Department of Tolima, Republic of Colombia, by Willard Ide Pierce, New York City.

Western Kentucky Coals and Cokes, by Joseph H. Allen, Mannington, Ky.

An Improved System of Water-Supply for Hydraulic Mining, by H. D. Pearsall, London, England.

The Bedded Ore-Deposit of Red Mountain, Colorado, by G. E. Kedzie, Ouray, Col.

The Russell Process in its Practical Application and Economical Results, by Ellsworth Daggett, Salt Lake City, Utah.

Certain Conditions in the Manufacture of Steel Rails which May Greatly Influence their Life in Service, by Frederic A. Delano, Chicago, Ill

Trough-Lixiviation, by Ottokar Hofmann, Alameda, Cal.

Note on a New Device for Operating Blast-Furnace Charging-Bells, by F. C. Roberts, Philadelphia, Pa.

A Method for the Estimation of Manganese in Steel, by Frank Julian, Chicago, Ill.

A Glossary of Furnace-Terms in English, French, and German, by Thomas Egleston, New York City

The Blake System of Fine Crushing and Its Economic Results, by Theodore A. Blake, New Haven, Conn.

Concentrating Magnetites with the Conkling Jig at Lyon Mountain, New York, by Ferdinand S. Ruttmann, New York City.

Modes of Occurrence of Pyrites in Bituminous Coal-Beds, by Amos P. Brown, Pittsburgh, Pa.

Notes on the Topography and Geology of Western North Carolina, —the Hiawassee Valley, by H. E. Colton, Chattanooga, Tenn.

The following resolution was unanimously adopted:

Resolved, That the Secretary be and is hereby instructed to express to the several individuals and corporations whose courtesy and hospitality have added so much to the pleasure and instruction derived from this meeting, the cordial thanks of the Institute, and to congratulate the Local Committee upon the complete success of its arrangements for the comfort and convenience of visiting members, a success which we, as engineers, are fully aware could not have been accomplished without enthusiastic and unwearied labor, organized and directed by admirable executive skill.

The meeting was then adjourned.

Excursions and Entertainments.

At the close of the first session a supper was given at the Hotel Brunswick to members and guests of the Institute by the Local Committee.

On Wednesday evening the subscription dinner took place at the Hotel Brunswick.

On Thursday an excursion party left Boston at 8.55 A.M. by special train on the Fitchburg Railroad, and visited the works of the Simonds Rolling Machine Co., the Putnam Machine Co., and the Fitchburg Steam-Engine Co., at Fitchburg, Mass. Luncheon was provided at the Simonds Works. On their return the party divided, some visiting the works of the Waltham Watch Co. at Waltham, and some the United States Testing-Machine at the Watertown Arsenal, all reaching Boston at 5 P.M.

On Friday a large number of excursions were made by separate parties to the following, among other places: the Clapp-Griffiths Steel-Works, at Wareham; the Lawrence Cotton-Mills; Tyler's Tube-Mills; the Merril Air-Engine of the Downer Oil Co.; the Chestnut Hill Pumping-Station; the Blake Pump-Works; the Golden Gate Concentrator; the Boston Heating Co.; the Sturtevant Mill; Harvard University; the Works of the Thompson-Houston Electric Co., at Lynn; the Boston Sewage-Pumping Engine; the Norway Iron-Works; the South Boston Iron-Works; the Ames

Shovel-Works; the laboratories of the Institute of Technology; the Boston Art Museum, and the collections of the Boston Natural History Society.

The Art Club, the St. Botolph's Club, the Electric Club, and the Boston Society of Civil Engineers extended the courtesies of their rooms to visiting members.

The manager of the Long-Distance Telephone placed an instrument in the Hotel Brunswick for the use of members in free communication with Philadelphia, New York, Brooklyn, and other cities.

The ladies accompanying members in attendance at this meeting were entertained at afternoon tea on Wednesday by Mrs. Francis A. Walker, and on Thursday by Mrs. William Rogers.

Members and Associates Present.

The following members and associates were registered at the headquarters:

Abbott, A. V.
Appleton, E. C.
Archibald, C.
Ashburner, C. A.
Atkins, W.
Bayles, J. C.
Bell, R.
Bigelow, A. S.
Bildt, C. W.
Billings, G. H.
Binsse, H. L.
Bowden, J. H.
Bowker, C. H.
Burden, H. H.
Butler, C.
Chance, H. M.
Clark, F. W.
Cogswell, W. B.
Coxe, W. E. C.
Coxe, E. B.
Crooker, R., Jr.
Daggett, E.
Daniels, F. H.
Day, D. T.
De Saulles, A. B.
Disston, T. S.
Downs, W. F.
Drown, T. M.
Duff, J.
Durfee, W. F.
Egleston, T.
Ely, E. B.
Emmerton, F. A.
Emery, C. E.
Emmons, S. F.
Eustis, W. E. C.
Ferguson, E. M.
Finch, I. A.
Firmstone, F.
Fuller, H. T.
Garrison, F. L.
Hall, E. J., Jr.
Hodge, J. M.
Hodges, A. D., Jr.
Holloway, J. F.
Hooper, W.
Hotchkiss, J.
Howe, H. M.
Hunt, A. E.
Hunt, W. P., Jr.
Johnson, J. E.
Kent, W.
Kirchhoff, C., Jr.
Kunhardt, W. B.
Lane, J. S.
Leckie, R. G.
Lewald, E. A.
Lewis, J. F.

Lodge, R. W.
Lukens, J.
Lyle, D. A.
Markham, A.
Moen, P. W.
McLennan, J. S.
Michaelis, O. E.
Morris, H. G.
Moore, P. N.
Munroe, H. S.
Neilson, W. G.
Newell, F. H.
Ostberg, P.
Parker, O. B.
Parsons, C. O.
Patch, W. B.
Payne, C. Q.
Pope, T. E.
Presbrey, O. S.
Price, J. A.
Pumpelly, R.
Raymond, R. W.
Rea, W. H.
Richards, R. H.
Robinson, T. W.
Rothwell, R. P.
Salom, P. G.
Scaife, W. L.
Sears, E. H.
Shed, N. W.
Shinn, W. P.
Smith, F. L.
Smith, H. S.
Spies, A.
Stearns, G. H.
Sykes, J. H.
Torrey, D.
Towne, L. O.
Walker, J. A.
Weeks, J. D.
Wellman, S. T.
Wheelock, J.
Wiley, W. H.
Wilson, E. B.
Williams, D.
Williams, O.
Williamson, J. J.
Woodward, H. G.

PAPERS.

GOLD AND SILVER MINING IN UTAH.

BY O. J. HOLLISTER, SALT LAKE CITY.

(Utah and Montana Meeting, July, 1887.)

ORES of the precious metals were known to exist in Utah from about 1863, when Gen. Connor, having established Fort Douglas on the bench east of this city, scattered his men out over the Territory as prospectors. They discovered and located mines at Stockton, in Bingham cañon, in Little Cottonwood cañon, at Pioche, Nev., and, I believe, in other localities. Capital was interested, and attempts were made to smelt the lead-silver ores near Stockton about twenty years ago (1866–1867), but the conditions were unfavorable and the net result was failure. Thereupon the business was abandoned. Two or three prospectors did a little work in 1868 on what was afterwards known as the "Emma" in Little Cottonwood cañon.

In May, 1869, the Union Pacific and the Central Pacific joined the track of the first Pacific railroad upon the Promontory range north of Great Salt Lake. The same year, a branch-road was constructed from the main line at Ogden to this city. In 1870 the shaft of the prospectors on the Emma dropped into a body of high-grade argentiferous galena at a depth of ninety feet. A few cars shipped out late in the season advertised Utah as a new mining field. Prospectors rushed in from Nevada and elsewhere, and before the end of 1871 the streets of this city were thronged with wagon-trains bringing ores from all directions, even from districts two hundred miles distant, for shipment by rail. The greater part of these ores, however, came from the two ranges of mountains—the Wasatch and the Oquirrh—which one sees from this place, walling in the Jordan valley on the east and the west, and shedding their waters into the Jordan river.

From the beginning of 1871 to the beginning of 1887, sixteen years, the total output of Utah, inclusive of the metals in the ores sent elsewhere for reduction, was 326,000 tons of crude lead, 13,476 tons of refined lead, 59,065,016 ounces of fine silver, 135,135 ounces of gold, 4332 tons of copper matte, and 343 tons of copper pigs; worth in the aggregate (rating crude lead at $60 a ton, refined lead

at $120 per ton, fine silver at $1.12½ per ounce, gold at $20 per ounce, and copper matte and pigs at 12½ cents per pound) $91,497,058. The actual value, as calculated from year to year, is within $100,000 of this sum. The average per year for sixteen years is about five and a half millions; for the last ten years it is $6,868,562, the precious metals being three-fourths of the total value.

As these sixteen years have passed the silver miner has seen his ounce of fine silver fall off in market value from about $1.30 to 95 cents. The loss to him has been in a measure made up by the increase of facilities, by the improvement in appliances, by the introduction and use of better methods, by gain in skill through experience in carrying on his business.

Utah ores are mainly lead-silver, the notable exceptions being the sandstone ores of the Silver Reef district and those of the Ontario group. An ever larger and larger proportion of the latter, however, are sent to the smelters rather than to the chloridizing mill. The annual output of ores for the last ten years I should place at about 120,000 tons. For the year 1884 I looked it up with care for the Director of the Mint, and found it to be 149,062 tons: 43,451 tons from Beaver county, 22,944 tons from Juab county, 41,069 tons from Summit county, 26,811 tons from Salt Lake county, 2905 tons from Tooele county, and 12,882 tons from Washington county. About the end of 1884, the Horn Silver Mine in Beaver county ceased the heavy production it had then kept up for a number of years. This would have reduced the annual output by 40,000 tons but that a simultaneous improvement in the price of lead stimulated the working of other mines and thus in part made up the loss.

A glance at a map of Utah will show the wide distribution of our mining districts. Beaver county lies about 200 miles south of Salt Lake, Summit, and Tooele counties, while Washington is 100 miles south of Beaver, and Juab is not far from half-way between Beaver and Salt Lake.

Ores of good quality have been found in a score of isolated mountains rising out of the desert of western Utah like reefs or islands out of the sea, but any considerable development of mining in that region must doubtless await additional facilities, especially those of transportation.

In the Wasatch range the mining fields, present and prospective, are at the points of greatest upheaval—where the granite appears bearing up on its rugged shoulders the quartzites and dolomites; where, in a word, the range is highest and most massive.

North of this point, so far as the Wasatch continues, indications of the precious metals are occasionally observable, but nothing has been found as yet of practical importance.

The great district of the Wasatch, which has produced one-third of Utah's total output, and which those who have most studied it believe to be still comparatively unknown and unexploited, lies a little south of east of this city, where the Uintah cross-range abuts on the Wasatch. There on the heads of the two Cottonwood cañons, of the American Fork, and of streams which find their way into the Weber and the Provo, the granite has raised up a field of dolomite 8 or 10 miles square—possibly many more—breaking and twisting and tilting it up at all angles in the process. The waters and the glaciers have left but small patches of it now, sticking in the peaks and high ridges between the diverging streams. But in or near these are found the Emma, the Flagstaff, the Joab Lawrence, the City Rock, the Prince of Wales, the Miller, the Pittsburg, the Eclipse, the Reed & Benson, the Maxfield, the Crescent, the Apex, the Anchor, the Daly, the Ontario, and many other mines and prospective mines.

Eighty miles south of this great mining field, in the same range, Mount Nebo rises to a height of 12,000 feet, and here the silver-lead veins occur again. They are said to be thin and close, and the ores to be poor in silver; but this reputation may be found in time to be based upon the lack of any real knowledge of their character.

South of Mount Nebo the range becomes a trough for the Sanpitch river, which flows southward through Sanpete valley to a junction with the Sevier river. The Sevier comes from the south in a similar trough-like depression in the range, and after receiving the Sanpitch finds its way out upon the desert.

About the sources of the Sevier the range again becomes massive, as at Nebo; and east of Beaver, and in the neighborhood of Mount Baldy, near Beaver, we have the lead-silver veins again. A long way from railroads, they have not as yet been profitably wrought.

Beyond this district, called Marysvale, to the Rio Colorado, the range is of a later growth, or has come up from shallower depths. Where the Colorado cañon passes through it, the overlying sandstone is a mile thick. The mines in that part of Utah occur in a sandstone reef, which faces the Wasatch at a little distance for a hundred miles.

Mines are found on both slopes of the Oquirrh range, at an elevation of 5000 to 7000 feet, from Great Salt Lake southward 70 or

80 miles, as at Stockton, Dry Cañon, Ophir, Bingham, and Tintic. There the range subsides, losing itself in the general level of the country. Beyond the Sevier it reappears, but is soon lost again, a lava plain and the desert intervening. It reappears for the third and last time between Milford and Beaver as the Granite range, the southern section of which is a diamond-shaped reef of dolomite, 6 or 8 miles long by 3 or 4 broad at the broadest point, rising, as it were, out of a sea of eruptive rock. South of Beaver river, the place of the Oquirrh seems to be taken by the sandstone reef before spoken of.

Volcanic action is very marked from Fillmore south into Iron county, where the famous iron reefs and "blow-outs" somehow resulted, and for a hundred miles beyond them. About half-way between Beaver and Fillmore, the subterranean fires caused by the displacement when the Wasatch was raised, have scarcely cooled. In a sulphur basin half a mile in diameter, near Cove Creek Fort, one may stand in a cut less than twenty feet deep and hear the cauldron boiling beneath his feet with unpleasant distinctness. Nature is still retorting sulphur from the still hot depths, and the crust has barely become solid. Near by, a volcano rears its extinct crater 800 feet above Pine creek. The lava-flow all but buries it on the east. To the west it presents a cone that can be seen 30 miles away. There are other extinct craters in the neighborhood, and toward Fillmore, and other beds of sulphur, widely distributed, still hot at slight depths.

The Frisco mines are in the San Francisco range twenty to thirty miles west of the Granite range. Eighty miles south of Frisco is the Tustagubit, a limestone mountain full of soft low-grade silver-ore, occurring in caves. Some 40 miles west of Frisco is the Wah Wah range, in which promising prospects have been found and considerable work has been done.

I have taken this cursory view of our mining districts to give some idea of their wide distribution. There is not a county in the Territory where the prospector has not left his footprints. The maps of ten years ago showed 80 mining districts, comprising 1500 square miles. Wherever there are mountains, unless they are built up of sandstone (and these are not always an exception) mineral indications are abundant, and valuable mineral deposits or veins are likely to be found in time.

And well it is that this is so, for while the depletion of a mine must necessarily result in that mine's exhaustion, in so extensive a

mineral field, mining will practically never cease. New mines are ever being discovered in old districts, and new districts promise to be brought under requisition for a thousand years to come. Methods will be bettered, processes cheapened, appliances improved, and thus the waste dumps of to-day be made the mines of to-morrow.

It is well, I say, for these Rocky Mountain Territories and States that this is true; for their agricultural and pastoral capabilities are restricted by their topography and climatic conditions within comparatively narrow limits. All of their business—manufacturing, railroading, trading, husbandry—is based upon mining, and mining of the precious metals at that.

During the fifteen years when the output of Colorado was stationary at about $3,000,000, her population was stationary at about 30,000 souls. As her output of the precious metals increased, in the same ratio all her other business increased. With an output of $30,000,000, she counts her population at 300,000; her mountains are crossed hither and thither and her cañons threaded by railroads; her hamlets have become towns, and her capital aspires, not without reason, to a place in the first rank among great towns.

Let me now pass our mines briefly in review by counties, beginning with

Beaver County.—About 17 miles west of and 1700 feet above Milford, a contact of trachyte and dolomite strikes north and south along the east base of the Grampian mountain. The principal mine of Beaver county, so far as known, is an immense ore-chimney in this contact. Early in 1876 it passed from the hands of the discoverers into those of Campbell, Cullen & Co., and in February, 1879, was by them sold to the Horn Silver Mining Co. The new company took a one-fourth interest in the extension of the Utah Central Railway from Juab to the mine, 140 miles. The extension was completed in 1879–1880. They established refining works in Chicago, and in 1881 built 5 smelting stacks within sight of this city, thereby reducing the cost of smelting, which had been before carried on at Frisco, near the mine, from $29.41 to $14.73 per ton.

The total output of the mine to the end of 1884, when production comparatively ceased, was 204,607 tons of ore. The yield of this ore was 69,380 tons of lead bullion and 7,260,566 ounces of fine silver. The lead and silver sold for $13,190,838, of which $4,000,000 was disbursed in dividends.

There is no water in the mine, and but very little in the vicinity. During the year 1884 it cost $5.07 per ton to place the ore on the

cars. Cost of supplies, inclusive of 1,481,000 feet of timber, was $57,942.23. There were 27,000 days' work on ore, 52,524 pit cars hoisted, 3600 railway cars of ore shipped, 427 tanks of water used. Cost of transporting ore to smelter was $225,000. The smelter reduced 103,079 tons of material. Cost of smelting was $13.29 per ton of ore; per ton of bullion, $38; refining the same in Chicago, $8. More than 100,000 feet of timber per month went into the mine, yet it cost but 50 cents per ton of ore to timber.

The walls of this great ore-chimney have come close together twice, in a vertical depth of 1200 feet, dividing it into three almost distinct ore bodies. It has been found impossible to support the trachyte-hanging country, which disintegrates upon exposure to the air by the extraction of the ore; and two caves have occurred in the history of the mine, too late, however, to do much harm.

About the end of 1884 the mine had been newly equipped with power to go to a depth of 2000 feet. A new shaft had been sunk, well out in the hanging country, and connected with the 7th level of the old shaft by drift—equivalent to the 10th level of Discovery shaft. Preparations had been made to drive different levels the entire length of the ground, 2940 feet.

Of the present condition of the mine I am not prepared to speak. It is understood that the 13th level has been reached and opened, and that there is a great deal of ore which, from its poverty or baseness, or both, cannot be profitably taken out at present. Occasional shipments are made, however, and it is very improbable that the history of this great mine is a tale that is told.

If the contact-fissure in which it occurs contains but one ore-pipe in its course of 2 or 3 miles, it will be strange. The showing is promising both north and south of the Horn Silver ground, and in both directions prospecting has been done, resulting, however, in failure. Either the prospectors became discouraged too easily, or their operations were ill-advised.

The Carbonate and the Rattler are locations on a strong fissure in trachyte striking at right angles with the Horn Silver fissure. The vein-filling is chiefly decomposed trachyte, soft and light, mixed with ore in the ore-chimneys in the proportion of four parts of trachyte to one part of ore. From 4 to 8 tons of this material are concentrated into one by the Krom process at a cost of $1.50 a ton, six men dressing 30 tons a day of ten hours. The product is half lead and contains 60 to 100 ounces silver per ton. The mines are opened to a depth of 600 feet, and furnish their own water.

The Cave property comprises 200 acres on the western slope of Cave mountain, a section of the Granite range 7 miles southeast of Milford. The formation of stratified blue and white dolomite, dipping into the mountain at an angle of 45°, has been twisted and broken, vertically, and along this break there is a series of caves, containing ores (sand carbonates and limonites) bearing gold and silver, fine fluxing material, worth on an average perhaps $25 per ton. The workings extend into the mountain nearly 2000 feet, and vertically 1000 feet. These properties belong to the Frisco Mining and Smelting Co., which is not doing much with them at present.

The mountains and hills bordering Beaver River valley in the vicinity of Milford were the scene of great activity in the early days of Utah mining. The surface bonanzas gave employment to several mills and smelters, although there was then no railroad. The surface deposits exhausted, the energetic men of those times soon found the country too slow for them and sought other fields. Men for the most part of small means have stuck to these mines, however, shipping a little ore every season and thus keeping the pot boiling, patiently awaiting the time when capital shall seek them.

The country is dry and forbidding in the summer, but there are water and wood enough, and out-door operations are never interfered with by snow or cold weather. There are ores of all grades and varieties, in large quantity and small. They are wagoned to Milford from Lincoln, and Star, and Bradshaw districts, 10 to 15 miles, and carried by rail to this city for 6 or 7 dollars a ton.

Boxelder County.—The country west and northwest of Great Salt Lake to the limits of the Territory is in Boxelder county. There are several mining districts in that part of the county, but, as I have said, their development to any considerable extent awaits the building of railroads to get in and out upon.

Juab County.—Tintic is the principal district of Juab county, and a very interesting district it is. There are supposed to be three main ore-channels, located and worked under various names from 1 to 3 miles in length, each of them, about a half a mile apart, and striking north and south.

The Eureka Hill group is the great mine of one of these ore-channels, and of the district, so far as known. I think it safe to say that the Eureka Hill mine has turned out 50,000 tons of $40 ore in the last four years. The Beck & Bullion group is a good second to the Eureka Hill. The formation at these mines is strati-

fied blue limestone, standing vertically on edge, and striking north and south. The ore makes between the strata, appearing to have replaced the limestone. Sometimes it is the thickness of one stratum, sometimes of several strata, and it occasionally makes across the stratification, so that the openings present a labyrinth now 600 to 900 feet deep, 1800 feet long, and the full width of the location—300 feet. The ore-bearing formation is, indeed, much wider than 300 feet; it is, perhaps, 600 feet. Comparatively dry, these ores have found a more ready market in Colorado than in Utah. These mines are fairly well equipped, have still a great deal of unexploited ground and large ore-reserves. The Eureka Hill is said to have a fine body of good ore at a depth of 700 feet from the surface, the deepest working, I believe, in the district. There are many promising locations about Eureka Hill, but depending mainly on their product for development, they are slow in becoming important producing mines.

The Crismon-Mammoth group, a mile and a half south of the Eureka Hill, has an ore-chimney of mammoth proportions, but the ores bear silver, gold, and copper in about equal degrees, and so have presented a difficult problem to their owners. All processes of reduction have been successively tried and abandoned—the gold mill, chloridizing, chlorination, lixiviation, smelting. A process has been tried of late on some 200 tons of the ore by which it is claimed that $15 ores can be profitably treated. The mine has practically an unlimited quantity of ores of that grade, with a fair proportion that are three times as good.

Half a mile east of this ore-channel, and southward, is a second ore-channel, located as the Sunbeam, the Elmer Ray, the Kenzie, the Hidden Treasure, the North Star, the Dragon (iron), the Carisa, the Northern Spy, the Spy No. 2, etc. Ore has been recently found on a location in line with and a mile north of these, supposed to be on the same ore-channel; which, if true, would show it to be three miles long. It is generally a contact between limestone and iron. The smelters near this city have for years procured their iron-ore for fluxing in the neighborhood of the Dragon iron-mine. Of the locations named, most of the surface bonanzas were exhausted at an insignificant depth. The Northern Spy, worked in a moderate way, has paid for a 10-stamp chloridizing mill, $30,000 in dividends, $32,000 for adjoining ground, and is said to have $100,000 worth of ore in reserve ready for stoping.

Half a mile east of this ore-channel, and further south again, is a

third ore-channel, located as the Julian Lane, the Silver Bell, the Golden Treasure, the Tesora, the Aspinwall, the Jo Bowers, etc. The outcrop on many of these locations is immense, almost covering their entire surface and containing rich ores in pots and chimneys, presenting extraordinary inducements to the experienced miner with money at his command, but sure to remain relatively unproductive and of course unknown until such miners take hold of them.

Ores of lead, copper, gold and silver are found to the westward of Tintic, for a distance of 50 miles; wherever, indeed, there are mountains or even hills. All that can now be said of them is that the conditions are not such as to permit of their being profitably wrought.

Salt Lake County.—The mines of Salt Lake county are at Bingham cañon in the Oquirrh, and on the Little and the Big Cottonwoods opposite, in the Wasatch. The ores of Bingham cañon are principally lead-silver, and, since the backs of the veins or deposits were worked out, comparatively poor in silver, low in lead, and base, the mass of them requiring concentration before they can be marketed.

The main ore-channel starts near the top of the range and strikes northeastward to the valley, three miles, crossing upper Bingham cañon, Bear gulch, Yosemite gulch, and Copper gulch, varying from 12 to 190 feet in thickness—a contact between quartzite foot and lime-shale hanging country. The ore makes in pipes 100 to 150 feet through on the course of the ledge, and from 2 to 20 feet in thickness. The ledge pitches northwest about 45°, and the ore pipes pitch in the vein toward the southwest.

The ledge is located as the Nast, the Saturn, the Utah, the Jordan, the Spanish, the Old Telegraph, the Brooklyn, the Miner's Dream, the Wasatch, etc., and within fifteen years vast quantities of ore have been taken from it, chiefly out of the ridges between the gulches where surface agencies had oxidized and concentrated the ore.

The Old Telegraph, a consolidation of twenty-one locations, covering a mile of the ledge, under a former ownership turned out 60,000 to 70,000 tons of ore, which sold for $1,500,000. The Jordan group is a mile long, and is crossed diagonally by the Galena, which, with the American Flag, and the Excelsior, belong to the property. At the intersection of the Galena and the Jordan 100,000 tons of lead-silver ores were formerly taken out, which sold for about $2,000,000, and there is said to be now lying there a million tons of

$20 quartz, in which gold and silver are so combined that it has as yet been found impossible to work it to advantage.

The Brooklyn lies east of the Old Telegraph and covers half a mile of the ledge. Its yearly output now is not far from 10,000 tons. The Yosemite and the Lead mines are on a similar ledge 20 to 30 rods distant from the ledge I have been talking of. They have been, and still are, large producers, with the moiety of their ground yet untouched. The Lead is said to have a breast of clean shipping-ore 30 feet thick on its lowest (7th) level. Pretty well down toward the valley, and having a southern exposure, the Brooklyn, the Yosemite, the Yosemite No. 2, the Lead, the Miner's Dream, and the Wasatch, have not reached the iron pyrites, lean in both lead and silver, upon which the deposits of these great ore-channels higher up and with a northern exposure, have bottomed. Their product is principally carbonates and sulphates. The shipping-ores contain about 10 ounces of silver, and 40 to 50 per cent. lead. 20 tons of the low-grade ores are cheaply dressed to 4 or 5, the gain being in lead rather than in silver. Such ores were worthless ten years ago, and there can be no doubt, I think, that millions of tons of Bingham ores which are now considered worthless will in like manner be profitably handled a few years hence.

There are, of course, many valuable mines in Bingham which are not on this principal ore-channel. The Winamuck and the Dixon, the first mines encountered on entering the cañon—they are immediately at the railroad station and open on a level with the creek—have produced above water level some $1,500,000, the ores averaging 60 ounces per ton, three times the average of the run of Bingham ores, even those of the surface.

There are vast deposits of quartz on Carr Fork, similar to that of the Jordan, bearing both gold and silver. A gold-mill does not catch the gold closely in this material, and it is too lean to justify costly processes of reduction, and so it lies bleaching in the sun and leaching in the rain.

The South Galena, the Live Yankee, the Aladdin, the Live Pine, the Silver Shield, the Last Chance, the Lucky Boy, and many other mines, 50 or 60 in all, help to swell the output of the district. Some of these are first-class mines, but more than half of them ship less than 100 tons each per year. They are worked by "leasers," and necessarily in a small way.

The Cottonwood cañons ship about 2000 tons of ore in a season, chiefly from half a dozen mines. The great lode is the Emma, of

dolomitic lime 150 to 250 feet thick, pitching into the mountain at an angle of 45, known in these days as the New Emma, the Joab Lawrence, the Flagstaff, and the Eclipse. The Emma and the Flagstaff ceased their enormous production 10 or 12 years ago, but prospecting from tunnels for new ore-bodies at greater depths has been carried on ever since, with only occasional interruptions. The productive stage of the Eclipse is still ahead, and the Joab Lawrence is a steady producer.

On the ridge above the Emma ledge are the Prince of Wales, the City Rock, the Evergreen, etc., which are regular shippers of ore. Below the town of Alta, on the long steep slopes facing each other, operations on various mines seem never to cease or to amount to much. Among the reasons are lack of means and short seasons.

The Emma belt extends across the Big Cottonwood as the Reed & Benson, the Kessler, the Silver Mountain, and the Maxfield. Besides these there are a hundred more or less promising locations on the Big Cottonwood.

Summit County.—Uintah district in Summit county is the name of that part of perhaps the greatest mineral field in Utah which lies east of the dividing ridge at the head of the Cottonwoods, and contains the Ontario, the Daly, the Crescent, etc.

The Ontario ledge, inclusive of the Daly, is opened continuously for a distance of 6000 feet. The main working shaft rests at present at the 10th level, there being some years' work for the mill above that level. From this ground about $20,000,000 has been taken in the last twelve years, nearly half of which has been disbursed in dividends. Eastward the vein is interrupted by an outburst of porphyry, but westward the openings and the appearances all favor the belief that it extends to the divide—two or three miles, with perhaps one or two similar veins accompanying it.

The Uintah is a wet district; there is a heavy drift; and mines cannot be opened without the use of money to start with. Lack of money has kept the district back, but the extensive exploitation of the Ontario vein and of the adjacent country by the Ontario and the Daly companies have been so suggestive of the great possibilities of the ground on their general course westward, that on the strength of these developments, taken together with the surface indications, money is at last forthcoming to exploit this ground, although still somewhat timidly. The locations are being combined in groups and gathered into the hands of parties able to improve them. The new shaft of the Anchor is said to have cut a vein between its 5th and

6th levels as rich and strong as that of the Ontario. This shaft is half a mile west of the Daly shaft. The Anchor is troubled with water, and is at this moment advertising for bids for the excavation of a drain tunnel more than a mile long which will draw off the water to a depth of 1100 feet.

The Ontario is a small vein in quartzite, but its average output for ten years has been about 20,000 tons of dry ore per year—the ore shrinks 25 per cent. in drying—worth in round numbers $100 a ton. The mine has paid its 132d dividend of 50 cents per share —64 on 100,000 shares, 68 on 150,000 shares—in all, $8,300,000, $66 on the old shares, $34 on the new shares.

Four hundred and fifty men are employed at the mine and mill, at an average wage of $3.50 a day. The mine consumes 15,000 tons of coal in a year, 400,000 feet of lumber, 200,000 running feet of round timber, 45,000 pieces of lagging. The mill reduces 65 tons a day, using 15 tons of coal and 15 to 20 cords of wood, the latter in the dryers and Stetefeldt furnaces. Coal costs $4.50 a ton, wood $5 a cord, lumber $25 per M., round timber 6 cents a foot, lagging 6 cents each. Cost of mining per ton is about $12, of milling $14, of hauling, prospecting, dead work, repairs and incidental expenses $5,—a total of $31.

Great difficulties have been met and conquered in the history of the Ontario. The arrangements for taking care of the water are now ample (for some years at least), the mine is well equipped, all branches of the business are systematized, the mill process approaches perfection, and the life of the mine has been lengthened by a generation through the acquirement of adjacent ground. The mine probably sustains, directly and indirectly, 3000 or 4000 souls.

There are other productive mines in the district, not on what is now supposed to be the line of the Ontario. The Crescent, two miles west and 2000 feet higher, has an ore-channel 1200 feet long and 20 rods wide lying about 100 feet under the surface of a long steep hill, and sloping with the hill toward the northwest. It is not very unlike the deposit in Fryer Hill, Col., except in the grade of its ores, two-thirds of which are too poor to ship without concentration. The Apex is a companion mine to the Crescent, lying adjoining and contiguous, and presenting many of the same characteristics. The Crescent is now sinking a shaft on a perpendicular vein at the inner end of a 2000-foot tunnel driven from the face of the hill under the ore-deposit. In the same vicinity are the Sampson and the Boss, large veins lying under and partly in Piñon Hill, and containing bunches of high grade argentiferous galena.

Out of all this ore bearing region the water might be drained to a depth of from 2000 to 4000 feet by a tunnel striking the Ontario main shaft at the 14th level, three miles from the entrance underground of the tunnel. The course of the tunnel would be near and along the eastern extension of the Ontario vein, or what is believed to be such, for a distance of two miles, and in these days of machine drills and high explosives its excavation would be a comparatively inexpensive undertaking. An outlay of $10,000 a month for three years would probably accomplish it, working on a single heading. Along its course are the Hawkeye, the McHenry, the Lowell, and the Parley's Park, upon each of which shafts have been put down 300 to 400 feet, and levels and crosscuts and adits driven, and then the drivers themselves driven out by underground waterspouts. The Ontario could probably excavate this tunnel at less expense than it will cost to sink its main shaft from the 10th to the 14th level.

I think there are possibilities in Uintah district worthy of the attention of mining engineers. Although it has turned out over $20,000,000, it is without doubt still in the earlier stages of development and production. It contains no Comstock, but its veins and deposits are of a character more profitable to exploit and to work as a whole than the vast barren Comstock ledge with its two chief bonanzas, one for each mile of its length and for each fourteen years of its life.

Tooele County.—The mines of Tooele county are on Ophir and Dry cañons, opposite Bingham cañon on the western slope of the Oquirrh, and near Stockton, between Ophir and Great Salt Lake, in the foothills. One goes out there in an hour via the Utah and Nevada Railway, rounding the point of the mountain by the lake shore.

The country at Stockton is quartzite and lime, underlaid by syenite. Granitic porphyry dykes disturb and cross the veins, which strike mainly with the formation. The gangue is oxide of iron, quartz, spar and clay. The ore is silver-lead, mostly carbonate, free from base metals and very desirable as a flux.

The leading mine is the Honerine. The ore occurs in well-defined chimneys, of which there may be half a dozen in the course of 1000 linear feet. The workings strike water 800 to 900 feet from the surface. Four-fifths of the ore requires concentration, which is done in a mill near the mine. The shipping-ore and the concentrates average perhaps 30 ounces silver and 60 per cent. lead. Until recently, when the smelters materially increased their work-

ing charge on carbonate ores, the Honerine was selling in this market 400 tons per month. The output is somewhat less now. The mine has paid nearly $100,000 in dividends, is pretty well equipped, and has 10,000 tons of ore ready for stoping in reserve.

A dozen or twenty mines at Stockton, at Dry cañon, and at Ophir, similar in character and in nature and quality of their ores to the Honerine, are wrought with more or less success, according to the means and the enterprise and energy of their owners.

The output of Tooele county might as well be 30,000 or 50,000 tons of ore per annum as 3000 or 5000; and this is true of the mines on the American Fork in Utah county, which used to be heavy shippers but are so no longer. "Assessment work" never makes mines.

Washington County.—In this county, the Silver Reef mines turn out about 250,000 ounces of silver per year, chiefly from the properties of the Christy and the Stormont companies. The ores contain perhaps 20 ounces of silver per ton, one-fourth of which is profit, speaking roughly. For the past ten years the output of Silver Reef has been in the neighborhood of 3,000,000 ounces. The workings of the two companies named are quite extensive, and they have a good deal of unexploited ground. There is no reason to suppose that this silver-bearing sandstone reef, which is 100 miles long, contains ores rich enough to pay only in the locality where they have so far been wrought.

This ends my review by counties. I have aimed to give a general idea of the subject without encumbering it with details. I think all Utah men conversant with the facts would agree with me in saying that as respects Utah mining the harvest is great and the laborers are few. The output of Utah might be made equal to that of Colorado by the judicious application of money to the business. With more eastern and local railroads, which the coming years are certain to bring us, a rush of people to Utah may confidently be expected, with money to use and with sense enough to ascertain where it can be used to the best advantage. There is not much scope for agriculture in this Territory. If it is to support more than about a quarter of a million inhabitants, the mining foundation of its industries must be greatly broadened. It certainly can be. Nature has done her part. The mines are here. The mountains are full of mineral from Franklin to St. George, from the Wasatch to the Sierras.

Some fifteen years since, a mining craze having been worked up

on the extraordinary output of the Emma and other newly-discovered mines, many Utah mines were sold abroad or in the East at extremely high prices, but the money for the most part went into the hands of promoters and middle-men. Very little of it was ever expended in improvements or in mine development in Utah. I believe the business of mining has had to depend more upon itself in Utah, and that it has been conducted more on the profits or product realized, than in any other State or Territory. Hence, when the surface bonanzas were exhausted, many good mines were virtually abandoned. In some cases new funds have been got hold of, or a long siege without adequate means, depending altogether on product, has enabled owners to strike the second and even the third bonanzas.

The Winamuck and Dixon, mentioned before, is an instance of abandonment at water-level, after an output of 25,000 tons of 60-ounce ore. Almost all the mines of Bingham, now all together turning out 50,000 tons a year, were long since thought to be exhausted.

At Stockton the Honorine is a resuscitated mine, and so are all the mines now worked in Tooele county.

At one time the Ontario ore ran only $67 a ton for a year. A one-fourth interest in the mine was offered for sale at $375,000. The mine was carefully examined and the offer declined. That one-fourth-interest has since earned for its owners in dividends five times $375,000.

The Eureka Hill mine was offered for sale and partly sold in the East at what were regarded as extortionate figures. Somehow the sale miscarried; local owners after a time again got hold of it, carefully resumed work, carrying it on with the proceeds, shipping some seasons hardly twenty car-loads of ore. But in the last four years the yield of the mine has been enormous, and as yet there are no signs of exhaustion.

When the Horn Silver Mine was sold by Campbell, Cullen & Co., all the experts could see 500,000 tons of ore in it. When 50,000 tons had been extracted that body of ore was all gone, but two distinct ore bodies of unsual dimensions have been since found below it.

The Crescent, in Summit county, was virtually abandoned as the Piñon for ten years, and then a hundred thousand tons of ore were found in the hill.

The point I seek to impress is that constantly there has been a lack of means on the part of Utah miners to employ in extensive appliances and deep and wide working. It is ten years since the enor-

mous output of the Emma and the Flagstaff ceased. Exploitation has been carried on ever since in both of them under unusual natural difficulties, without satisfactory result. With plenty of means, justifying enlarged plans and more push, and enabling work to be done to better advantage, it is likely that new and even larger ore deposits than those found near the surface would ere now have been disclosed. These two mines turned out in their earlier years seven or eight million dollars.

Aside from the mills at Silver Reef, there are in Utah the Northern Spy chloridizing mill at Tintic, and the Ontario and the Daly chloridizing mills at Park City. There are two or three gold mills, now idle. There are three smelters—the Germania, the Hanauer, and the Mingo, running two or three stacks each, and the Horn Silver smelter, with five stacks, the latter idle at present.

Perhaps twenty per cent. of our ores are bought by the smelters East and West, competition between them forcing them to offer prices for certain kinds of ores which the Utah smelters decline to give.

The reduction of the cost of coke to six or seven dollars per ton—it is now twice that—and additional smelters in this valley, would of itself largely increase the output of our mines, and these conditions will be secured by the construction of additional railroads from the East to this valley. This is being done at this moment, so that the outlook for mining in this Territory and in districts naturally tributary to this valley may safely be regarded as promising.

ORE- AND MATTE-ROASTING IN UTAH.

BY RICHARD H. TERHUNE, SALT LAKE CITY, UTAH.

(Utah and Montana Meeting, July, 1887.)

IT cannot be said that the development of processes for roasting in Utah has been evolutionary. Some of the best systems were contemporaneous with smelting here, and one of the most primitive methods, pile roasting, is still entrenched, to the exclusion of all others, at one works. This is certainly not because we have not had sulphur, in place and out of place, to contend with, but rather because roasting has not been a necessity, we having, like most other smelting districts in America, a good share of oxidized ores.

But everywhere the increasing scarcity of carbonates, and even sulphates of lead, is stimulating inquiry about the different systems of getting rid of sulphur. If what I have to say about the past and present status of roasting sulphurets in Utah affords any useful hints towards the more economic solution of the problem, the purpose of my paper will be fulfilled.

In 1876 Mr. A. W. Geist put up and used a small Brückner cylinder at the Flagstaff works at Sandy. It had a firebox mounted upon wheels and movable over a small track, to facilitate barring off matted material and afford quick control of air-admission. This cylinder was connected with a reverberatory hearth above its level, for utilizing waste heat and gases; from this hearth the partially prepared charge was raked into an inclined chute which conveyed it into the cylinder below. The movable fire-box and hearth seem to have been good features, though they have not been followed. Of what this furnace did, and how well it did it, there is no record. It was idle for years after the works were abandoned, and was finally purchased by the French Company for experimental use at the Old Telegraph works.

The next Brückner furnace was put in by the Germania Company in 1886. It is a wrought-iron shell, 18 feet long, with a maximum diameter of 7 feet. It has a 4-inch lining of common brick, shaped in moulding to the curvature of the cylinder. It is driven by gear, friction-rolls having failed with this furnace, owing to the weight and talus of the ore, with an excessive moment of resistance. The best result claimed for this furnace has been the roasting of eight tons of matte (the weight of charge) in forty-eight hours to 4 to 6 per cent. of sulphur; equally favorable results have been obtained in treating pyritiferous ore containing 38 to 35 per cent. of sulphur. One man was employed on the twelve-hour shift to run the furnace. For further economy of labor, as well as for increase of capacity, two more cylinders were added to this plant this year. They are each 22 by 7 feet. The three are driven by a small special engine from an overhead counter-shaft provided with coned pulleys and with loose pulley on the driven cone, thus admitting of change of speed and independent stoppage of any cylinder. The new furnaces were lengthened for heat-economy as well as gain of capacity. This hope has been fulfilled, but, owing to imperfect draught, the work of all three is far below the standard of the original one. Mr. James, the present superintendent, believes the economic limit of length has been reached in the 22-foot cylinders. The credit of their adoption is due to the late superintendent,

Mr. T. S. Austin, who also ingeniously diverted the waste heat through a revolving dryer. He found that where the moisture in base ores was even as low as 3 per cent., crushing in rolls and screening through No. 4 wire screen was very much quickened by previous drying. No difficulty is found in discharging the roasted ore through four man-holes, two on each side, three and a half feet between centers. About 20 per cent. of coal is used, from the Pleasant Valley coal mines. Two men per shift of twelve hours attend to the three roasters. The expenditure of power to turn them is very slight. Some difficulty from friction attends the use of worm-gear which drives the main pinion. The first furnace had no worm-gear when first built.

Regarding lead and silver losses the company is somewhat reticent. In ores carrying 20 per cent. of lead, which they have successfully treated, they admit large lead-losses from volatilization, silver losses being moderate. They claim the plant is too new to have found out just what it can do. Taken in connection with the compact automatic crushing mill it is an interesting addition to the metallurgy of the Territory, and reflects credit upon the management for their liberal investment.

During the closing months of Mr. Brückner's life, he studied and patented different combinations of two cylinders for the perfect utilization of heat. He proposed also a slagging hearth, which would remove one great objection to the cylinder principle, by matting the charge, but he did not live to see either feature adopted. After a careful study of such combinations, fired from one fire-box, I would reject them as too costly; trouble in one furnace would of necessity stop both, which would mean too much idle plant.

The Hoffman furnace, which is a Brückner cylinder with a fire-box at both ends, has had no application in Utah. Unless the furnace be longer than would be practicable for other reasons, there is no apparent advantage from the fire at both ends. Regarding the continuous-feed rotating cylinders, they have not been used here for purely oxidizing furnaces. They are looked upon with favor by mechanical engineers but not by metallurgists.

A revolving horizontal hearth, with dome-roof and fixed rabbles, has been used at the Revere mill, Bingham, but with what results it is not possible to learn. The works are abandoned: a pile of roasted ore there indicates that desulphurization was very complete.

Reverberatories have always been used in Utah. They are all of moderate length, with low arches. There are none as yet with a distinct fusion-hearth. They offer the advantages, here as elsewhere,

of moderate lead and silver losses and of the capacity of treating ore high in lead, yet too base to work raw. Such ores exist in enormous quantities in Bingham. A notable deposit has recently been exposed in the Lead mine, of which several hundred-ton lots, removed in developing, assay 41 galena, 7 blende, 31 iron pyrites and 21 silica.

As 50 per cent. of these ore bodies is too fine for the pile-roasting, the treatment of the screenings in reverberatory furnaces becomes the only alternative. Working them raw in the blast-furnace, while possible, where oxidized ores abound, and while carried on to a large degree here, is yet not rational, since a ton of this ore will produce 1400 pounds of iron matte. This matte, commonly from three to six times as rich in silver as the original ore, ties up a large amount of capital, and must itself be roasted and freed from a sulphur content as great as that of the ore from which it was made. The further details of this problem, showing just why prior roasting of this ore is proper, belong to a paper on smelting. Suffice it to say that the company owning the ore is building a large battery of reverberatory furnaces, including one furnace with fusion-hearth, for its treatment. These will be 59 feet 8 inches long in the hearth and 13 feet interior width, and without the expensive return-flue under the hearth, since the ore is of low grade and the flue-dust of slight value.

Roasting in the different types of kilns has had only an experimental existence in Utah. It was done by Mr. Knapp at Sandy in 1879, with the open-top Styrian kiln; by Mr. Wartenweiler, with arched kilns, in Bingham, about 1878; and by Mr. Hesse, at West Jordan, in 1880, in a high kiln about six feet square and with grates at the bottom. At the Germania kilns have also been tried, but at the present time they are not used in this Territory.

In February, 1886, a rare opportunity was afforded by the courtesy of Mr. R. C. Chambers and the Daly Mining Company for testing the Stetefeldt furnace as an oxidizer of pyrites and iron matte. The experiment was conducted by Mr. T. S. Austin and the writer, aided by Mr. Joseph E. Galigher, superintendent of the mill. The new 30-stamp Marsac mill at Park City was tendered for the occasion at a time when the bins were full of rich ore, the mill complete in every detail, shifts of men organized for running it and stockholders eager for a start. Under these circumstances the experiments were somewhat hurried and incomplete, but important and hitherto unpublished data were obtained. The furnace was 6 feet

square and 50 feet high. A car-load of Dixon ore was first treated, having the following composition: 32.5 sulphur, 25.7 iron, 16.1 lead (fire assay), 15.9 silica, and 1.5 zinc, and 13.5 ounces silver to the ton, being a low-grade pyritiferous galena. It was passed through the driers and stamps (having No. 30 screens) and the furnace in five and a quarter hours. It screened at the battery without difficulty. The furnace was quite hot, and the scintillating burning ore as it descended the main stack of the furnace, was a beautiful spectacle. The mill was kept running, so that our attempts to properly separate the roasted material under many conditions of air-admission and temperature, were not satisfactory, and the variations in sulphur contents not marked sufficiently to deduce any law of oxidation in this furnace. The following are some of our results:

1. Main stack, early part of run, side openings, 15.8 per cent. sulphur in ore.

2. Hotter furnace, central discharge open and both holes closed, 6.2 per cent. sulphur left in ore.

3. Chamber dust, 5 per cent. sulphur left in ore.

4. Side openings only, last of run, 4 per cent. sulphur in ore.

5. Faster run, return flue, last of operation, 8.3 per cent. sulphur left.

6. Mill sample of entire run which we could extract from chambers and furnace, 7 per cent. sulphur, 17.5 lead, 15.75 silver.

Ten tons of iron matte of the following composition was provided for treatment: 22 sulphur, 13 lead, 6 copper, 55 iron, 59.5 ounces silver per ton. Of this amount only three tons were treated, owing to its obstruction of the battery screens and trouble with the feeding apparatus of the furnace, caused by heat in that region. The mill-sample of the roasted matte showed that its sulphur was half oxidized in the two seconds of time of descent. The roasted matte was necessarily contaminated with the pyrites in the conveyors. The following are assays under different conditions of treatment:

	Lead. Per cent.	Oz. silver.
1. Main stack, early part of run, side openings, 10.5 sulphur left in matte, ...	11.8	34.9
2. Lower hopper only open, 11.23 sulphur left in matte, ...	10½	52.48
3. Open hopper, last of run with low fire, 14.16 sulphur left in matte, ...	9.7	62.96
4. Final run, closed hopper, open sides, 12.38 sulphur left in matte, ...	10	62.40
5. Return flue and chamber dust mixed, 3.61 sulphur,	19.2	25.64
6. Mill-sample of all matte produced, 11.5 sulphur, .	12.1	46

Between adhesion to side-walls and loss of dust in the chamber and flue, which we could not recover, accuracy of data as to extent of oxidation and loss by volatilization could not be obtained. The particles of both matte and pyrites were rounded by fusion, and the mass of both were somewhat fritted together. Crude as this experiment of necessity was, it proved that the Stetefeldt furnace, made higher and with screen further from return-flue, to prevent its destruction by sulphur fumes, might be made available for cheap and rapid roasting of pyrites. Barring the needed fineness of the ore, the result with this method was quite satisfactory, and one of our smelting proprietors is hoping for an early trial on a more extended scale with this furnace. I would add that during this operation we fired with wood, and most of the time in the main and auxiliary fire-boxes.

Pile-roasting of ore and matte is still carried on extensively in Utah. Though offensive to man and beast and destructive to vegetation, it commends itself to the metallurgist by cheapness, low losses and favorable coarse condition of the material for the subsequent smelting operation. It is the one process in the whole realm of our profession about which there seems to be a veritable "fetich.' In it, to speak briefly, like conditions, as far as human observation and control go, rarely produce like results. We cannot grasp the cunning agencies which change matte and pyrites in pile-roasting into artificial hematite, with a kernel of cupric-sulphide, or into an aggregation of hollow shells, without metamorphosis. Countless have been the sizes and shapes of piles of these sulphurets, and the quantities and arrangements of wood employed, yet to-day no man can give us a formula for the process. The common practice is with pyramidal piles 24 by 18 feet at their base, 6 feet high, and fired with from 6 to 15 inches of light wood. Such a pile will contain about 80 tons of matte or pyrites, and will burn from 30 to 40 days. The material is broken to first size, and should be handled with sluice-forks to avoid fines. A heavy layer of wood will cause a thicker zone of fused raw material, but will often result in a very excellent roast; a pile recently burned with 6 inches of wood gave 3 inches of friable fused matte and an equally good roast.

Piñon pine will not do for the operation as well as lighter woods, such as white pine and quaking asp, balsam and even red pine. It produces too hot a fire. The assay of one of these heavy bottoms showed it to contain .70 arsenic, 5.00 insoluble residue, 36.20 iron, 8.10 copper, 25.00 sulphur and 20.00 lead. It was thus nearly

twice as high in lead as the original matte, being a sort of concentration by fusion, which, to my mind, has been suggestive. A more striking instance of this concentration by fusion was in the case of a roast-heap of pyrites containing 20 per cent. of lead, from the Spanish mines. This was raised in the fused mass to 48 per cent. of lead, and the silver was doubled, being raised from 6 to 12 ounces per ton.

The presence of copper in increasing quantity of late years in our ores, and, of course, in the matte, has made the latter more fusible. Now since disulphide of copper is much less fusible than protosulphide of iron, the greater fusibility of these mixed sulphides in matte can only be accounted for, as we do for the fusibility of polybasic slags, compared with those which are monobasic; that is, that the multiplication of bases increases fusibility in mattes, as it does in silicates. In the absence of data, I found by experiment that a 400-grain mixture of equal parts of Anaconda matte (70 per cent. copper) and the commercial FeS fused in a Hessian crucible in coke-fire in five minutes, while 400 grains of FeS required seven minutes and an equal weight of the copper matte did not fuse in fifteen minutes, when it was removed from the fire.

The cost of pile-roasting 5134 tons of roasted matte at the Hanauer works was $2.25 per ton, distributed as follows:

Labor, breaking with sledges,	$ .25
Teaming matte to roast-yard,	.30
Labor, building, dressing and breaking down piles,	1.10
Wood, ¼th cord,	.60
	$2.25

The selected roasted matte usually contains 6 per cent. of sulphur, being a three-quarter roast. The pile-roasting of ore that is coarse enough gives equally good results, and sometimes better. The sulphur in 82 tons of ore recently tested was reduced from 27.6 to 6.57 per cent. in the roast in a single operation. This pile yielded 86 per cent. of roast, and 14 per cent. of raw strippings which had to be re-treated. The cost of the single roasting was 60 cents per ton.

THE OLD TELEGRAPH MINE, UTAH.

BY G. LAVAGNINO, SALT LAKE CITY.

(Utah and Montana Meeting, July, 1887.)

THE old Telegraph mine lies on both sides of Bear gulch, a short branch of Upper Bingham cañon, and nearly in the center of the group of mines called the Bingham Mines, about twenty-seven miles southwest of Salt Lake City. It is considered by local miners as the most important mine of the district.

The geological formation consists of Weber quartzites, with subordinate conformable limestones and shales, alternating with porphyritic rocks. These widen towards and connect with the massive backbone of the Oquirrh mountains, while the quartzite develops itself on the slope broken through by Bingham cañon.

The porphyry not only runs between the strata, but also cuts across them, isolating large portions near the mountain mass, and again on a smaller scale it penetrates some joints in the quartzite. It is a crystalline felspathic rock, containing variable proportions of quartz, mica, hornblende, chlorite, and iron pyrites as accessories. In places, it takes a less distinct or almost uniform fine-grained structure. It has a gray or greenish, and, when much altered, also a reddish or even white color.

The quartzite is a fine-grained or almost compact grayish-white or bluish or reddish quartz-rock, containing in places fine scales of mica and grains of iron pyrites as accessories. The raised, folded, or broken masses show well-defined bedding-joints, and their general dip is 35 to 50 degrees towards north or northwest.

In the porphyry occur metalliferous veins with a prevailing N.E.-S.W. strike. These are in general narrow veins; their principal contents are not dissimilar from those of the other deposits; but there is considerably less oxidation near the surface and the ore is mostly richer in silver and gold.

In the quartzite the ore-deposits generally occur on the joints of bedding. The more important deposits occur either in clearly-defined contact-veins or, more often, in somewhat vaguely-defined mineralized zones, that could be called contact-zones. These comprise several masses of more or less altered or decomposed rocks and the newly-distributed products of their mechanical and chemical decomposition, such as breccia and clay in seams or in more irregular bodies.

These zones are more definitely limited beneath by a bed of hard quartzite upon which they rest, than on the opposite side by limestone or porphyry. White fine-grained limestone remains near the surface in isolated benches only, and in the upper part of the old Telegraph mine it is almost absent. Porphyry makes a more continuous limit, not only as a roof, but also often terminally.

The ore-bodies in these zones are very irregular. Although they are generally lenticular masses lying flat with the strata, they occur also in different forms and positions, and often they are connected by narrow channels, veins, or threads. When an ore-body, as in the more frequent case, occurs in soft ground, there is often in immediate contact upon it a local slickenside, commonly called a hanging-wall; but similar polished joints occur also through the ore-body itself. The country-rock is not always clearly distinct from the ore proper; more frequently it is more or less impregnated next to the ore, or even transformed so far as to be worth mining as ore. This is especially the case in the upper parts of the mines, in the regions of the oxidized ores, which have been hitherto the more important in this district.

In a very wide sense, the more important group of mines, extending from the Wasatch and Lead mines on the east, to the Jordan and Stewart mines on the west, can be considered as belonging to a single mineralized zone or belt, although portions of it may appear isolated by porphyry as above described, or by the later phenomena of erosion, which play an important part in this district, as attested by the existence of placers and of conglomerates in the cañon and in the gulches. Very likely, all these deposits have a common genesis with those of the old Telegraph mine.

At this mine there are large irregular bodies of more or less siliceous pyrites, transformed within in the limits of oxidation into more or less ochreous, spongy, and brittle quartz, and into more or less siliceous yellow, red, or black ochres. In or near these pyrite bodies, and also wholly apart from them, occur smaller bodies of galena, with fine-grained zinc-blende and iron pyrites, corresponding to the bodies of carbonated lead-ore within the zone of oxidation, which have furnished the main production of this mine since its discovery.

The distribution of silver and gold is very irregular; the main part of the ore is of low grade, while in smaller pockets the yield is considerably higher. The pyrite is of very low grade, that most free from quartz containing only 5 or 6 ounces of silver and less

than $1 gold per ton, although better samples both in silver and in gold can be obtained. In places it carries a little copper. The pyrite occurs both crystallized and massive. The "base ore," generally coarse-grained crystalline galena, with fine-grained zinc-blende, contains about twice as much silver as the pyrite.

The more or less spongy quartz above mentioned (the old matrix of pyrite) contains always a little silver and gold, mostly little more than the pyrite. In some places it carries 10 to 20 ounces of silver per ton, and in exceptional spots much more. Most of the silver and gold is contained in the small amount of ochre adhering to and coloring the quartz. Perfectly white spongy quartz is poorer; some of the best quartz is gray. The more or less siliceous and argillaceous ochre, usually extending into the country-rock, is in general of no practical value, unless it passes over into lead-ore proper.

The lead carbonate-ore is a more or less siliceous, ferruginous, and argillaceous lead-ore, colored yellow, red, light or dark gray, containing as accessories pieces of "free" galena—that is, galena free from other sulphurets. It carries mostly 10 to 12 ounces of silver and about $1 gold per ton. This yield is often raised by the presence in the ore-body of small veins of soft quartz, richer in silver and gold.

The above-mentioned types of ore are not always clearly distinct. There are transitions; and the country-rock next to the ore may be more or less impregnated with iron oxide and lead carbonate. The practical distinctions are between first-class lead-ore or smelting-ore, second-class lead-ore or concentrating-ore, and silver- and gold-ore. Pockets of true silver- and gold-ore (a brittle sandy quartz, with white or yellow clay) occur indeed in position even independent from the lead-ore, and appear to be a more recent production in the upper levels of the mine.

The old Telegraph mine was first discovered by prospecting near the top of the ridge, east of the gulch, on the outcrop of some bodies of lead-ore; and it has been opened and worked by adits or tunnels on both sides of the gulch; most of them originally driven to prospect separate mining claims. The gulch represents a large natural gap in the ore-bearing ground; it abuts on the quartzite south limit of this ground, and crosses a belt of porphyry north of it.

The more extensive and more productive workings have been so far on the east side of the gulch. The tunnels in part cross the formation and in part follow it along seams usually marked by clay or by more altered rock. Some of them are driven apparently at ran-

dom. Branches cross towards the south in the quartzite, which is here harder and little altered. Similar prospecting has been done by inclined or vertical shafts.

Almost all the tunnels have reached some lead-ore bodies. Raises and inclines, or more irregular openings, have followed the ore and eventually connected the several levels.

The connection between some of the ore-bodies has been proved in several cases, and in one particular instance this connection has been traced for several hundred feet, between the ore-bodies discovered near the top of the hill on the east and other ore-bodies discovered by lower tunnels. A similar relation exists on the west side of the gulch.

The ground is generally soft and needs timbering. The largest stopes have been timbered by "square sets." Most of the openings cave, unless their timbering is expensively kept in repair. This has only been done for the more important or more easily maintained entries and connections. Of the old stopes, only a few in hard ground or protected by a solid hanging-wall are open at present.

The importance of the work done from the beginning (1870 ?) to the present day can be best judged by the following round figures, which I give as my own imperfect estimation :

About 30,000 linear feet of more or less regular openings, such as drifts and shafts, have been excavated, of which about three-fourths is drifts and one-fourth shafts ; and again, about three-fourths is on the east side and one-fourth on the west side of the gulch ; and about 40,000 cubic yards of widening lateral to these openings, or of stoping proper, has been done in addition.

This work and the extraction of the materials did not offer any serious difficulty. Only the timbering has been expensive. A plant has been built for sawing and framing timbers. This plant includes one air-compressor, by which artificial ventilation has been provided in some of the workings.

All this work has produced, I presume, not far from 100,000 tons of oxidized ore. The first-class ore has been either smelted at the old Telegraph furnaces in West Jordan, near the Jordan river, about ten miles south of Salt Lake City, or, as during the last seven or eight years, it has been sold on the market. The second-class lead-ore has been concentrated at the old Telegraph mill in West Jordan, or at a smaller mill in the cañon. Some of this ore has been first lixiviated by filtration with hyposulphite of soda, to extract a small portion of the silver, and then concentrated. Some good and

some poor silver-ore has been lixiviated with hyposulphite of soda in a small mill at the mines.

The mechanical concentration presents two difficulties, which are especially notable when the ore is higher in silver and gold; namely, the siliceous tailings carry off several ounces of silver per ton, and the slimes much more. These slimes usually contain more silver and gold than the concentrated ore; and they cannot be successfully concentrated. They hold most of the ochre contained in the ore, and with a little care they can be kept free from the very fine sand which tends to follow them. But for their fineness, they would be a desirable smelting-material. The plan adopted when all the works were running (that is, over seven years ago) was to collect the slimes and to combine with them the flue-dust and make bricks ready for smelting.

The leaching with hyposulphite of soda meets with the difficulties due to imperfect filtration, to the presence in the ore of carbonate of lead, and to the variable proportion of silver contained as chloride.

It is unnecessary to give in this place any details about the old works, most of which are at present in a dilapidated condition; but I will briefly conclude about the mine.

It has been the misfortune of this mine, as of so many others, to be the subject of fraudulent manipulations and deals. But considering the money actually expended in working it, the old Telegraph mine has been on the whole a fair paying mine. Even leaving out of consideration the first two or three years of intense activity, following the discovery of several large bodies of ore, and carried on at a rate far surpassing the permanent capacity of the mine, and referring only to the last seven or eight years, during which it has been worked on a smaller scale, and mostly by lessees, we find that it has paid reasonably well, and especially so under good market conditions. The ore put in the market, almost exclusively carbonate, has been, on the whole, a desirable smelting-ore, containing mostly between 30 and 50 per cent. lead, and between 10 and 12 ounces of silver, and not far from $1 gold per ton. The concentrated ore has been generally a little richer in lead and silver than the first-class ore. The price realized in the market has mostly varied between $20 and $30, and the expense for putting it on the market between $2.50 and $4.50 per ton.

Last year, the production was about 4600 tons of carbonate lead-ore, averaging about 33¼ per cent. lead, 13 ounces silver, and $1.80 gold per ton. It sold well, the average smelting-charge being only

$5.50 per ton, beside the usual deductions of 10 per cent. of the percentage in lead, 5 per cent. of the yield in silver, and about 20 per cent. of the value in gold. The total proceeds of this lead-ore were about $102,000. Moreover, nearly $3500 was realized from the sale of about 80 tons of silver- and gold-ore, containing about 40 ounces silver and 1½ ounce ($31.50) gold per ton. The total expenses for freight, sampling, etc., were about $15,000; and the total net profit has been over $25,000. Over 2000 tons of second-class ore, with 13 per cent. lead and 9 ounces silver, over 500 tons of quartz with 14 ounces of silver and $1.00 gold per ton, and over 500 tons of sulphuretted ore with nearly 10 ounces of silver and $1.00 gold per ton, are still on hand. These figures show the relative importance of the lead-ore and the silver- and gold-ores.

This year the conditions of the market are less favorable. Although only lead-ore carrying over 40 per cent. is at present offered in the market, the smelting-charges required are such as to curtail production. The product now comes partially from a new body of red carbonate-ore, lately discovered. Second-class ore, exposed in several places, cannot just at present be conveniently worked. I give these details to show how much this mine and most of the Bingham mines are affected by the conditions of the market.

The problem of the immediate future lies in the utilization of the low-grade pyrites and base ore; and low freight-rates and economical concentration are the first steps towards its solution.

At present, only a few mines, directly or indirectly connected with smelters, are shipping such ore, while other mines, producing carbonate-ore, get along with difficulty. But as this oxidized ore is bound to become steadily a less important factor, the future life of many mines will depend on the sulphuretted ore occurring at variable depths.

The old Telegraph mine itself cannot be said to have been opened in depth. There is a lower tunnel, to which no oxidized ores descend, and in which a large amount of pyrites and a smaller amount of base ore are exposed, while only a small amount of good galena has been produced from it. Another tunnel, 50 feet lower, crossing a belt of porphyry and a bed of quartzite, has been provisionally suspended in limestone, before reaching any ore-body. The fact that the sulphuretted ore could not be worked with profit, and the doubt, I presume, of finding better ore, and possibly the theory that these deposits were wholly superficial, have prevented the opening of this mine in depth.

But in the Brooklyn mine, east of the old Telegraph, where the ore lay on a well-defined contact-joint, good galena ore is found at a depth nearly 1000 feet lower than the lowest old Telegraph tunnel; and it is reasonable to admit that under better commercial and technical conditions the sulphuretted ore in the old Telegraph mine may offer a larger field for mining than is or has been offered by the oxidized ore, and that the mine may maintain also in this new field its present leading position.

Before closing this paper, I will add a few words about some experiments made a few years ago, at the instance of Mr. G. De la Bouglise, for leaching the so-called silver-quartz and the siliceous tailings produced by the mechanical concentration of the more argentiferous second-class lead-ore. It was admitted that large quantities of so-called "silver-quartz," with at least 15 to 20 ounces silver per ton, could be taken out of the mine at little expense, and that this ore, as well as the iron pyrites, could be got down from the mine to the Jordan river, about 17 miles, by tramway and railroad, for not over $1 per ton, since years before a rate of $1.25 had been given for second-class ore. Other economical features of the plan were the use of the pyrites for making acid, availability of salt from Salt Lake, and a very favorable position on the Jordan river. I am sorry to say, that some of these economical advantages having thus far failed to be realized, the plan has been for the present laid aside, although the technical results of the experiments were, on the whole, satisfactory. Mr. De la Bouglise has taken patents for the process.

From the old tailings, residues of the leaching and concentration made in 1878 to 1880, as above mentioned, not more than 70 per cent., but from some good silver-ore from the mine as much as 95 per cent. of the silver could be extracted.

In these experiments, the liquor employed was hot brine, acidulated with muriatic or sulphuric acid to such an extent as was called for especially by the little carbonate of lead present in the one, and by the degree of acidity desired in the final liquor, ready for the precipitation of the silver. The carbonate of lead appeared to occasion no inconvenience beyond that of augmenting the consumption of acid, the silver in the carbonate being readily dissolved, and the lead also passing mostly into solution as chloride. The iron and copper from the ore, passing into solution, improved the chloridizing and dissolving properties of the liquor. Some of the gold also goes into

solution, which fact, with the coloration of the liquor, indicates that somehow chlorine is liberated and is active in the process.

The solubility of the chloride of silver is augmented by the presence of some free acid in the final liquor. But it might happen that such liquor, if further used on a fresh batch of ore in such a way as to neutralize it, would lose its color and part of the silver previously in solution.

As the free acid would cause a large consumption of copper or iron, if these were used to precipitate the silver, granulated lead-bullion (from the old slag-dumps) was experimented with as a precipitant. In small experiments, where the surface of the lead was kept clear by strong agitation, most of the silver was readily precipitated, while the solution was losing its color, and before any copper began to precipitate. On a large scale, this operation could probably be conveniently effected in revolving wooden barrels. Most of the chloride of lead can be easily separated from the liquor by cooling, and could be reduced to spongy lead by metallic iron, and used over again as an effective precipitant of the silver.

This part of the process needs, however, to be tested by more experiments on a larger scale.

The practical difficulties encountered are mostly of a mechanical nature—for instance, the difficulty of filtration through the ore and the handling of the liquor. The most favorable results have been obtained by agitation of the ore with the liquor, kept hot with steam, and by subsequent decantation. It is reasonable to suppose that pumps could be got to handle the liquor satisfactorily, especially by avoiding the pumping of hot liquor. In the experimental plant built at West Jordan, near Bingham Junction, to avoid any interruption on account of pumps, the liquor was carried up and down an inclined plane by the use of a cable elevator. In this plant, moved by water-power, there are agitating tubs of 1000 to 1500 lbs. capacity, and also smaller ones for testing on 20-lb. to 30-lb. charges. There are also five 750-lb. stamps, and one pair of Cornish rolls, for wet and dry crushing, a few pieces of concentrating apparatus, and one Brückner cylinder, 12 feet long by 5 feet 6 inches diameter. A small acid-plant, consisting of two pyrite-burners and a single lead-chamber, 50 x 20 x 18 feet, followed by a lead drum, 40 x 4 x 4 feet, need to be enlarged in order to work economically.

Several other kinds of silver-ores, both oxidized and sulphuretted, and also copper-ores, have been tested by this acid-process with varying results. Perhaps the most interesting case is that of some zinc-

blende of the Raymond & Ely mine in South Nevada, which, according to Mr. Godbe's laboratory tests, can be leached very satisfactorily without preliminary roasting, the zincblende itself being, of course, insoluble.

THE SULPHUR-DEPOSITS OF SOUTHERN UTAH.

BY A. FABER DU FAUR, NEW YORK CITY.

(Utah and Montana Meeting, July, 1887.)

AT the request of some of the members to whom I have shown a collection of specimens of sulphur from southern Utah, I herewith present some notes relating to the mines and the method of extracting the sulphur from the ore. In preparing these notes I have borrowed from a paper by Prof. G. vom Rath, published in the *Neue Jahrbuch für Mineralogie*, vol. i., 1884, dated Salt Lake City, December, 1883.

The Cove Creek sulphur-bed, now worked by the Dickert & Myers Sulphur Co., is in Beaver county, near the line of Millard county. It was first discovered by Mr. Ferdinand Dickert in 1869; other beds in the same neighborhood were found by him from 1872 to 1875, making altogether 22 mining claims. The want of railroad communication prevented the early development of these claims; but in 1883 Mr. F. Dickert put up a plant for extracting sulphur from the ore, which proved a perfect success; and in 1885 the property was transferred to a stock company organized under the laws of Utah.

The railroad station nearest to the Cove Creek mines is Black Rock, on the Utah Central Railroad, 204 miles south from Salt Lake City, and about 26 miles west of the mines. The road from Black Rock to the mines passes through desert and lava-bed, to the road leading from Kanosh to Beaver, which latter road it joins near Cove Creek fort, a square stone enclosure, erected by the Mormons for protection against the Indians. Cove Creek fort is situated near the eastern limit of a plain bounded on the west at a distance of about 12 miles by the Mineral range, and on the east by the Sulphur mountains, forming part of the Tushar mountains. To the northwest the plain runs into the Sandy desert. About 3 miles from the fort rises a solitary hill about 1000 feet above the

plain, which Prof. vom Rath considers an extinct volcano, and to which he gave the name Mount Dickert.

There are in the neighborhood of Cove Creek fort several sulphur claims; for instance, the Mammoth, the Sulphur King, and the Excelsior. In the Mammoth claim, limestone and slate are impregnated with sulphur to such a degree that pieces as large as one foot in diameter consist almost exclusively of it, containing only fragments of the surrounding rock. Gypsum is likewise found as the product of the transformation of limestone. In the cavities, sulphur is found in small octahedral crystals. At the Sulphur King, decomposed andesite is impregnated with sulphur.

The Cove Creek mines are situated about 2 miles southeast of the fort and to the east of the Beaver road, in a small basin near the foot of Sulphur mountains, surrounded by low hills, with a narrow ravine opening in the west-northwest direction into the plain. The basin is about 6000 feet above the level of the sea, while the Sulphur mountains to the east rise about 2000 feet higher. The hills surrounding the basin consist mainly of andesite, partly also of a very light white trachyte, which is used as a building-stone at the sulphur works. Obsidian splinters are scattered over the surface of the hills.

As far as explored, the sulphur-bed extends at least 1800 feet by 1000 feet, and the quantity of sulphur contained therein was estimated by Prof. vom Rath, at a time when the bed was not as fully exposed as it now is, to be at least 1,300,000 tons.

A curved cut has been made through the sulphur-bed near its western end, exposing a vertical wall of rich yellow sulphur, 34 feet high. From this cut a track leads to the dump above the smelter, a short distance to the west. The sulphur extends up to the surface over part of the basin, but is mostly covered with sand or rather decomposed andesite. The surface of the deposit is wavy, giving the impression of an agitated mass gradually cooled. The sulphur is partly mixed with sand or gypsum. Most of it is of yellow color, while some of it is dark gray, and is called "black sulphur." In the cleavages and cavities, very beautiful crystals of sulphur are found. It would appear that large masses of sulphur were thrown to the surface, penetrating and enveloping the loose surface-material, while at the same time vapors and gases passed through the semi-liquid mass during the process of cooling. The deposits of pure sulphur partly resemble the so-called "virgin rock", which is found as a product of distillation in the sulphur-flower chambers, particu-

larly when distillation goes on too rapidly. Some also resemble the delicate crystals found on the walls of such chambers; others are like the crystals formed in slowly-cooled masses of sulphur. Gases escape in many places in the cut and in the prospect-holes, together with water holding salts in solution. At some points also a considerably elevated temperature is observed.

The plant for extracting the sulphur from the ore and for preparing it for the market consists of four steam-smelters, two grinding-mills, two retorts, and a chamber for making flowers of sulphur, and apparatus for making roll sulphur.

The capacity of the four smelters is 10 tons of crude sulphur per day; but so far the production has only reached about 1200 tons in one year.

The sulphur is extracted from the ore by liquation, the smelters being cast-iron cylinders, into which the ore is charged and steam admitted. The charge rests upon a cast-iron grating, supported by a steam-jacketed kettle, into which the sulphur runs and from which it is tapped. The ore is charged through a manhole at the top, while the residue, which is retained above the grating, is removed through a lateral manhole. The sulphur from the kettles is run into moulds.

The plant for flowers of sulphur was erected last winter.

The greater part of the sulphur from the smelters is ground fine and shipped in bags of 100 pounds; the flowers of sulphur are also packed in bags. During the year ending May, 1887, over 1100 tons of ground sulphur were sold.

The great obstacle to the full development of this new industry is the cost of bringing the product to the market; but it is hoped that within a short time a railroad will pass within a very short distance from the mine, whereby the great expense of hauling to Black Rock will be saved.

HISTORY OF THE ONTARIO MINE, PARK CITY, UTAH.

BY THOMAS J. ALMY, SALT LAKE CITY, UTAH.

(Utah and Montana Meeting, July, 1887.)

THE Ontario croppings were struck July 19th, 1872, by Herman Budden, who, together with his partners, at once began to prospect their claim. Within five weeks their development work enabled them to sell it for $30,000 to R. C. Chambers, who represented the

now famous firm of Haggin, Tevis & Hearst. The new owners continued the prospecting work with such energy that $180,000 were expended within the next two years—the cash returns being about $40,000 for ore sold.

In January, 1874, a corporation was formed, and in December, 1876, was re-incorporated, with a capital stock of the par value of $10,000,000, divided into 100,000 shares. In the meantime, the McHenry and the Marsac quartz-mills of Park City had been leased. About 16,000 tons of ore were crushed between March, 1875, and December, 1876, from which the gross yield was $900,000, although the wet-milling process in use permitted a saving of only 80 per cent. of the assay-value of the ore. In July, 1876, a dry-crushing mill, with forty stamps, was designed; and the work was pushed so rapidly that on January 10th, 1877, it was ready for the first trial, with all its machinery in motion. Built originally at a cost of about $250,000, it has since received additional new machinery, one new furnace, ore- and salt-houses, and various labor-saving appliances, swelling the cost of the plant to $360,000, but also making it complete in every respect.

Disaster came in October, 1878, when a fire destroyed extensive buildings and costly machinery. The direct and indirect loss amounted to $200,000. In forty days the pumps were again set in motion, and by January, 1879, the hoisting works over No. 1 shaft had been rebuilt, when operations proceeded as before.

In June, 1877, the Ontario stock was listed on the New York Stock Exchange at $20 per share. Up to the present time, the quotations have ranged considerably above this figure, once reaching $42 per share. Very little stock, however, changes hands, as it has always been held as an investment and not as a speculative property. The first dividend of fifty cents per share was declared in April, 1877. Besides several extras a regular monthly dividend of fifty cents per share has been paid ever since, excepting an interval of eight months; but during this period the amount of the passed dividends was expended for new property. In January, 1881, the capital stock was increased to 150,000 shares. The total number of dividends to June 15th, 1887, is 133, amounting to $8,375,000.

The magnitude of the surface improvements and underground workings can scarcely be appreciated by any one not familiar with all the details. Aside from prospecting, the problem of drainage has been the source of greatest extraneous expense. The mine has been worked through three shafts and a drain-tunnel since 1883. Prior

to that time, the abundant water was a continual trouble and heavy expense. In that year were completed: first, the drain-tunnel, starting 500 feet below the Ontario mill, with a length of 5765 feet, tapping the ore-body on the 600-foot level, and costing, with all the necessary adjunctive work, $200,000; and, second, a hoisting-works, with a Cornish pump, among the finest on the Pacific Coast. The hoist, pump, and foundations cost $690,000, and the buildings $62,000. The engine has 2500 horse-power, and the pump's capacity is 2200 gallons of water per minute from a depth of 1800 feet. The hoisting-works, with pumps and machinery, at Shafts Nos. 1 and 2 cost $95,000; while a village of necessary buildings at the mine and mill have added an expense of nearly $100,000. Other property, supplies, tools in use, swell the grand total of this class of expenditures to $2,570,000, in addition to the cost of the drain-tunnel. The area of ground stoped out to December 31st, 1886, is about 125,000 square yards. The length of exploratory and drainage tunnels, cross-cuts, etc., equals nearly five miles, while the total openings of all descriptions are estimated in linear measure at four times that length.

In addition to the outlay for prospecting (which amounted in 1886 to $4.31 per ton of ore extracted), the expenditures have been largely, but wisely, increased by purchases of adjoining ground.

There are steadily on the pay-rolls of the company about 400 men, at an expense of $40,000 per month; while indirectly its operations maintain probably 2000 people.

In 1886, the best ore was sold to smelters, and averaged $94.82 per ton. Ores of lower grade were milled, averaging 54.32 ounces of silver, with a saving of 91 per cent. The cost per ton of ore, including all expenses, has steadily decreased to $29.65 in 1886. Up to May 31st, 852 tons of bullion had been sold for $16,517,900, and ore-sales had netted $1,725,100.

The company carries a large reserve in the treasury, insuring the regularity of dividends against all ordinary mishaps.

Mr. R. C. Chambers has been manager of the Ontario from the day of its purchase.

SILVER-MINING AND MILLING AT BUTTE, MONTANA.

BY WM. P. BLAKE, NEW HAVEN, CONN.

(Utah and Montana Meeting, July, 1887.)

THIS camp is just now startled by the alarming suggestion that unless there shall speedily be an appreciation in the value of silver, and a decrease in the cost of salt, it will be prudent business policy to close the Alice mines and mills until the conditions of production are more favorable. Such an event would be a great blow to the prosperity of Butte. It would throw some hundreds of men out of employment, and it might lead to the adoption of the same policy by other companies, which, like the Alice, have not been able to pay dividends to their shareholders for a long time. The question arises, and becomes emphatic: If, under the present adverse conditions, the mines and mills cannot be made to pay a proper profit, why should they continue to be worked? The owners mentally ask: Shall we go on exhausting our mines, pouring out a stream of silver for the maintenance of the community, and the support of the railways, or shall we husband our reserves, withdraw the miners from our slopes, and wait for more favorable conditions? This is the impending question, and all concerned dread to meet and answer it.

Let us briefly review the situation and forecast, if possible, what the solution of the problem will be.

It must be understood that the discussion does not include, or refer to, the immense copper-producing interests of the camp, which may or may not be upon a very much more satisfactory basis. It will be confined chiefly to the operations of the Alice company, statistics of which are at hand, without involving the other silver companies in the conclusions beyond those inevitable from well-known facts, patent to all.

There are at present in Butte and in its vicinity in the principal mills an aggregate of 290 stamps dropping upon silver-ores:

Alice,	80 stamps.
Moulton,	40 "
Lexington,	50 "
Silver Bow,	30 "
Blue Bird,	90 "

The Alice mills at present, with their 80 stamps, devour, in round numbers, from 100 to 120 tons a day. The actual amount

crushed in the Alice mills, net, dry tons, from May 15th, 1880, to May 31st, 1887, is 207,391 tons, yielding a total product of $5,996,-720.53 in gold value; being an average of $28.91 per ton, as shown in detail, for each year, by the figures in the annual statements of the Alice company, from which the annexed table is compiled, except for the five months in 1887, which are from the company's books.

Value of Bullion produced from the Mines of the Alice Company, May 15th, 1880, to May 31st, 1887.

Period.	Assay Value @ 1.29[29] per oz.	Discount on Assay Value.	Net Value Realized.	Tons Ore Worked.
May 15, 1880, to Dec. 31, 1881,	1,193,425.45	153,280.48	1,040,144.97	30,498
Year 1882	904,970.71	128,637.30	776,333 41	23,382
" 1883	1,166,227.49	186,847.27	979,380.22	30,706
" 1884	1,239,651.13	194,408.88	1,045,242.25	33,201
" 1885	1,175,767.59	226,324.55	949,443.04	34,996
" 1886	1,166,267.37	286,066.69	880,200.68	39,739
Jan. to May, 1887	438,441.32	112,465.36	325,975.96	14,869
	7,284,751.06	1,288,030.53	5,996,720.53	207,391

The costs of mining at the Alice for the year 1886 were $7.44 per ton, and for milling $7.57 per ton, making $15.01 for mining and milling, irrespective of the proper allowances for dead work, permanent improvements, additions to plant and property, superintendence, etc.

It is claimed that the ores are worked to within 90 per cent. of the assay-contents of silver, and 40 per cent. of the gold-contents.

According to the official reports of the Lexington (French) company* for 1883 and 1884, the extraction at the Lexington mine amounted to 19,943 tons in 1883, and 22,551 tons in 1884, at an average cost of $12.54 in 1883, and $12.33 per ton in 1884. The yield per ton for 1883 was 265.70 francs, and 224.27 francs per ton in 1884, or respectively $53.14 and $44.85. The bullion varied in fineness from 550 to 800, and the weight was 22,132 kilos. fine silver and 266.150 kilos. of fine gold, for which 4,965,393.05 francs were

* *Societe Anonyme des Mines de Lexington—Rapport du Conseil d'administration à l'Assemblée Générale Ordinaire*, du 18 Juin, 1885.

realized. The additional sum of 89,001.45 francs from the ores treated at the old mill carried the production up to 5,054,394.50 francs, or approximately $1,010,878.90 in value for 1884. The cost of milling fell from $14.11 per ton in 1883 to $12.10 per ton in 1884. The percentage of precious metals saved was 86.10 for silver, and 52.30 for gold.

The observation is made that the low product of the ore for the last months of the year 1884 had continued through the months of January, February, and March of 1885, but it had improved somewhat in April and May. It is also stated that the direction had found it necessary to show less rigorous care in selecting the ore, and that a change in the dip of the vein had led to the poorer and sterile ores.

The later reports of this company are not at hand, but I am informed that the grade of the ores worked has not been maintained at the high figures reached in the years 1883 and 1884.

Later information, since this paper was presented at the meeting, shows that the total amount of ore treated at the Lexington from October, 1882, to June, 1887, was 110,981 tons, yielding bullion to the value of $4,014,769. The average quantity of ore crushed per month was 1950 tons; and the average product was $91\frac{7}{10}$ per cent. of the assay for the silver and $59\frac{5}{10}$ for the gold. The grade, or assay-value, of the ore, which at the close of the year 1884 was about 40 oz. silver, and 0.65 oz. gold, declined in 1885 to 32 oz. silver, and 0.5 oz. gold, and since then has declined to from 29 to 30 oz. silver and 0.37 oz. gold per ton.

The Lexington and Silver Bow mills are now running partly, and the Moulton chiefly, upon custom ores from various parts of the district, mostly from claims which are being prospected by the locators or owners.

The Blue Bird mill, of 90 stamps, 20 of which have just been started, is running upon the ores from the mine of that name, and is one of the most perfect mills of the West. The daily output from the mine has not been ascertained.

The statistics of the other companies are not at hand, but for the purposes of a general estimate of production it is fair to assume that the Lexington, the Moulton, and the Silver Bow companies have together worked one and three-eighths times as much ore as the Alice; and from the known fact that the silver ores of the district are nearly the same on an average in their silver contents, the total production of the Lexington, the Moulton, and the Silver Bow may be esti-

mated at $8,245,490.71, which sum, added to the Alice product, makes the total $14,242,211.24. I have left out of this estimate the product of the Blue Bird company, inasmuch as its mill was finished in the winter of 1886–87, and thus has not been running long; and there may also have been some other producers of a smaller amount of ore and bullion; but it is believed that the round figure of $15,000,000 of value would fully cover the whole production of silver and gold bullion in Butte and its vicinity since active silver mining and milling commenced.

The total amount of dividends by these companies publicly reported and recorded in the columns of the *Engineering and Mining Journal*, up to June 18th, 1887, was as follows:

Alice Company, ...	$750,000
Lexington Company, ...	565,000
Moulton Company, ...	350,000

Aggregating $1,665,000. Leaving out of the calculation such smaller amounts as may have been paid by the Silver Bow and other companies, of which no records exist or are accessible, it will be seen that the sum of dividends is but a little over 11 per cent. of the market-value of the product. If calculated upon the production of the Alice company alone, it is a little more than 12 per cent. of the product.

Of the total of fifteen millions of gross product, over thirteen millions of dollars, or about 90 per cent. of the whole, has been paid out in wages, supplies, machinery, salt, and transportation, and has largely contributed to the building up of the city of Butte, the Summit Valley mining district, the railway, and, in fact, the Territory of Montana.

If objection should be made that the gross product of the camp is over-estimated, whereby the ratio of dividends to gross products would be made to appear too small, the fact remains that neither the Alice company nor the Lexington has of late been able to declare any dividends from earnings. The last dividend of the Alice was in September, 1886, and the last of record paid by the Lexington was in January, 1885. The last dividend paid by the Moulton company, which completed the total of $350,000, was paid in May, 1887.

The present condition of the Alice, as respects production, grade of ore, and discount on silver, is shown by the following figures for the first five months of 1887, obtained from the company's books:

Product of the Alice Mines in Silver and Gold for the first five months of 1887.

Months.	Tons of Ore Worked. Tons.	Lbs.	No. of Bars of Bullion.	Gross Weight. Ounces.	Gold. Ounces.	Silver. Ounces.	Assay Value @ 1.29.29	Discount.	Amount Realized in Gold Dollars.
1887									
Jan'y.	2212	402	80	119,451.30	140.09	49,053.19	63,932.69	14,154.41	49,578.28
Feb'y.	2579	1168	92	138,439.90	178.19	57,986.51	78,658.33	17,841.28	60,817.65
March.	3373	1669	124	185,254.90	192.72	72,135.38	97,272.69	25,633.20	71,639.49
April.	3244	1031	124	190,459.50	216.83	70,914.94	96,193.47	26,122.50	70,050.97
May.	3459	32	150	230,500.10	186.43	76,397.93	102,604.14	28,713.97	73,890.17
	14,879	800	570	864,005.70	914.26	326,487.95	438,441.32	112,465.36	325,875.96

We are not to forget that the greater part of the working capital of the Alice has been derived from the product of the mines. The mines have paid for a very extensive and costly plant. Three double-compartment shafts have been sunk; one to 700, and another to 1000 feet, and some eighteen miles of cross-cuts and drifts have been run. The Alice shaft has one of the finest hoisting works in the West, with double-decked cages, direct-acting Corliss hoisting engines, one 12-inch Cornish pump, and mills, water-works and other permanent improvements have been added from time to time. Purchases have also been made of adjoining claims, especially of the Magna Charta and Valdemere, by which the mining territory and productive capacity of the company have been greatly increased. The outlays for these improvements and property should be counted with the dividends in estimating the outcome of the enterprise. The value of the surface plant January 1, 1887, was, in round numbers, $625,000, and the cost of the Magna Charta and Valdemere will amount to about $200,000; a much smaller sum than these added properties have yielded in net returns. The Magna Charta has alone for many years supplied at least one-half of the ores which have been run through the company's mills. It has been highly advantageous to the company to have several sources from which to draw its supply of ore.

There has of late been some adverse criticism of the management of the company because the reported bullion production seemed to warrant the resumption of dividends since January 1st, of this year. I say "reported," because it has been the habit of the local press of Butte to compute the value of the bars as shipped at one dollar per ounce, presuming the bullion to be 800 fine, as formerly, whereas it now ranges from 300 to 400 fine only. For example, the Alice

bullion shipments for May, 1887, were reported in one of the papers as aggregating 117 bars, valued at $196,726; when, in fact, there were 150 bars worth only $73,890.17—this being the amount realized for them in the market. The practice of reporting bullion-shipments by value alone, is decidedly reprehensible. The weight and fineness of the bars, or the number of ounces of fine silver and gold, should always be stated. We cannot otherwise have reliable statistics of the production of the precious metals.

We may find an explanation of the unprofitable condition of silver mining at Butte in three chief causes:

1.—The gradual exhaustion of the richer free-milling surface-ores, and the higher ratio in quantity of lower-grade ores now worked.

2.—The depreciation in the value of silver.

3.—The high price of salt, the cost of which, relatively to the product of bullion, increases rapidly as the value of the ore diminishes.

Exhaustion of Free-Milling Ores.—It has always been known that the admixture of the surface oxidized ores with the chloridized sulphuretted ores from below has given the best results in milling. In general, also the oxidized ores have been of higher grade than the unoxidized. These surface-ores, while not yet fully exhausted on the Magna Charta and Valdemere claims, are not as easily obtained of the desired grade as formerly; and the time is not distant when the milling will be confined wholly to the unoxidized ores from below. Although, as will be seen from an analysis of the figures given, the grade of the ore worked has constantly declined since 1880, it does not follow that the ores of the lode diminish in value as depth is attained. No careful investigation of this has been made, but appearances of the lode at the greatest depth attained are as favorable for rich bonanzas of ore as on the higher levels. The decrease in average value per ton, as worked, would seem to be due rather to the greatly increased quantity worked, demanding greater rapidity of extraction and less care in the selection. The quality is of necessity sacrificed to quantity. But this is in the right direction. Economy requires that methods of working shall be so cheapened that all the ore of the great bodies in the lode can be worked. It is poor economy to drive long exploring drifts and maintain the expensive plant in order merely to seek out the choice bodies of ore—to take the cream of the mine, "pick out its eyes," as the Cornish miner so well expresses it, and leave the great bulk of lower-grade ore behind.

Depreciation of Silver.—The depreciation of the value of silver is a serious loss to the silver industry, not only here but wherever silver-mines are worked. When the Alice company was organized, in 1880, fine silver sold for about $1.14 per ounce, and in July, 1886, it had fallen in value to 91 cents per ounce, a difference of 23 cents. The aggregate loss from the standard value of $1.29[29] is shown from year to year in the first table given, and amounts to $1,288,030.53, up to May 31st, 1887. For the first five months of 1887, the average selling-price of silver and the percentage of loss to the Alice company, ranged about as follows:

	Value per oz.	Loss, per cent.
January,	$1.02	22.21
February,	1.01¾	22.68
March,	.95	26.35
April,	.95¼	27.16
May,	.95	27.99

While a discussion of the silver problem will not be here attempted, attention is directed to the eloquence of these figures in favor of some possible combination of silver-producers by which the price of silver can be controlled and regulated, so as to maintain a higher and more stable ratio of value to gold.

Salt.—We now pass to a consideration of the high cost of salt as one of the chief factors in the problem of profitable working of the silver ores of the district.

Before the completion of the railway from Ogden, and for some time afterwards, salt cost the mills at Butte from $80 to $30, downwards to $23 per ton. This was during the period of working the higher grade ores at and near the surface and in comparatively small quantities. The advent of the railway and the final cheapening of salt was a great boon to the silver mines, and immediately permitted large bodies of ore to be worked that before the reduction it would not pay to handle.

The published statements of the Alice company show that since its organization up to December 31st, 1886, it has paid for salt to the Union Pacific Railroad Company the sum of $602,392.08. In order to compute the total consumption by the camp the same ratio applied in calculating the production may be used, viz.: one and three-eighths the Alice figures. This would give in round numbers a value of $828,000 for the salt used by the Lexington, Moulton and Silver Bow companies, or a total, including the Alice com-

pany, of $1,430,000 in value. Adding to this the estimated consumption for the first six months of 1887, and an allowance for the amount used by the new Blue Bird mill, we have, in round numbers, $1,550,000 for salt paid out by the camp to the Union Pacific Railway, over five-sixths of which was for freight alone.

About eighteen months ago the price of salt was reduced by the railroad to $15 per ton, which, with the added cartage of $1.75 per ton from the railway, makes it cost the mills $16.75. At the Alice mills 15 per cent. of salt is used, and its cost per ton of ore chloridized is $2.52. This is 5 per cent. of the value of fifty-dollar ore, and 10 per cent. of twenty-five-dollar ore, and it amounts to about 30 per cent. of the whole cost of milling, or more than the labor costs. The importance of this item is thus more clearly shown.

The salt-supply is controlled by the railroad. Its first cost at the Great Salt Lake is from $2 to $3 per ton. It is reckoned at $3 by the railway and the freight at $12. The consumers say that it can be had in quantity on the railway at $2 per ton, and that, if allowed a satisfactory freight rate, they could buy of the producers to better advantage than of the railroad.

It may be truly said that this item of salt is the only one susceptible of immediate and considerable modification. If the fate of silver-mining in Butte hangs upon it, a sagacious management of the business of transportation cannot fail to conclude that it would be wise to deliver salt at or near its cost and thus to stimulate the silver interest into renewed vigor and give it promise of life for years to come.

All the interests of Butte are bound together. The successful prosecution of silver-mining, and the promotion of the growth of Butte in population and wealth, react favorably upon the copper interest. Nature has bountifully filled this valley with deposits of copper, silver, and gold; the basin is ribbed with veins which have as yet been but superficially opened, and which promise a basis for mining and metallurgical development for generations to come.

ENGINEERING RELATIONS OF THE YELLOWSTONE PARK.

BY DR. THEODORE B. COMSTOCK, CHAMPAIGN, ILLINOIS.

(Utah and Montana Meeting, July, 1887.)

To the large majority of visitors the unique features of the National Park are interesting chiefly on account of their novelty. Scientists of all schools may find here food for reflection, and much that is rare and marvelous; but the real, practical uses of the reservation, aside from its pleasure-giving adaptations, are not as yet fully appreciated.

Since 1873, the writer has on frequent occasions expressed freely his own views upon the proper means of utilizing the Park for the best interests of science.* Some of the recommendations made have been adopted by the authorities.

At the suggestion of friends I present here a brief statement of facts concerning matters which may be of advantage for visiting engineers to know, hoping that the very limited stay of the Institute may thereby be rendered a little more profitable, and that our common profession may possibly receive some benefit.

First, then, it should be understood that, as in many other cases in the West, the present routes to the Park from the south are not the most direct. The expedition of Captain (now Major) Jones in 1873 discovered a practicable route for a wagon or railroad *via* the head of Wind River, through Tó-gwo-tee Pass and Two-Ocean Pass, by which the distance from Point of Rocks, Wyoming, on the Union Pacific R.R., to the Yellowstone Lake, is only 290 miles. Eventually, it is probable that a railway line from Cheyenne or some other point in Wyoming will be constructed, which, taking advantage of

* *Reconnaissance of N. W. Wyoming, including Yellowstone National Park*, by Captain W. A. Jones, 1873. Report of Theodore B. Comstock, Geologist of the Expedition: Government Printing Office, 1874. Fuller Edition, Office Chief of Engineers, U.S.A., 1875. Papers in *American Naturalist*, *American Journal Science*, on the geology of W. Wyoming, 1873–74. "Scientific Value of Yellowstone Park," *American Naturalist*, March, April, 1874, etc., etc. See also Report and Memorial of Committee of Amer. Assoc. Adv. Sci., presented to Congress 1878: Government Printing Office, 1878. This committee, composed of Professor Joseph Henry, Professor O. C. Marsh, Major J. W. Powell, Lieutenant George M. Wheeler, with the writer as Chairman, secured an appropriation of $10,000, which was expended in building roads through the Park by Supt. Norris.

the rich mineral and pastoral resources of the intervening country, will afford quite easy access to the geyser region to travellers from the East. At present the Northern Pacific R.R. has the best facilities for reaching the tract, although the U. P. R.R. has cut off much of the former wide détour by building the Utah Northern from Bryan, Wyoming, northwestward.

Directly east of Yellowstone Lake, high up in the Sierra Shoshone (Snake Mountains), lies the Stinking-Water Pass, discovered by Captain Jones, and south of this, in the same range, another pass occurs, but neither of these is well adapted for engineering purposes, the cañons of the Stinking-Water being very deep and narrow, and the approaches of the passes very steep.

Two-Ocean Pass, at the head of the Upper Yellowstone, southeast of Yellowstone Lake, is one of the most interesting geographical features in the world. This was recorded quite inaccurately upon old maps from statements of guides and trappers, but its existence was not authenticated until actually surveyed by Captain Jones's party in 1873. The first accurate accounts were given by Captain Jones and myself in 1873 in our reports in scientific journals, and the first explanation of its geological significance was given by myself the same year.* Two-Ocean Creek is a little mountain stream which passes down across a low tract of meadow, where it divides into two portions, the one forming Atlantic Creek, soon joining the Upper Yellowstone river, the other going westward as Pacific Creek, finally reaching the Pacific Ocean by way of the Snake and Columbia rivers. In riding through the Park several other very low divides may be observed, but none which so gradually part the waters of the two great oceans.

The topographical features all through this region will afford many surprises to engineers unfamiliar with the causes which have acted here, but I am confident that many of our western members will better understand some of their home contours after running over this ground, even hastily. I have come across many puzzles in Colorado geology, which my studies in the Park have helped me to unravel. As to vein-formation, there is here, as it were, an epitome of the whole subject, so far at least as large areas elsewhere are concerned. Having amplified this subject already,† no more need be added in this place.

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* *Am. Naturalist* and *Am. Journal of Science*, November, December, 1873.

† "Geology and Vein Structure of S.W. Colorado," *Transactions*, vol. xv., p. 218.

The Yellowstone Park is a natural zoological and botanical garden, owing to its peculiar situation. The headwaters of the Missouri, Colorado and Columbia rivers are all suckled here from the same breast of snow, so to speak. Each great stream, as it emerges from the uncommonly rich fountain-head, plunges through cañon walls into a desolate plain, across which only straggling remnants of nature's biological types pass from the fertile regions below. Here the moose and the bison, the grizzly bear and the mountain sheep, and many other almost extinct types have taken refuge from the relentless pursuit of man; animals and vegetation representing the most widely separated regions of North America have thus wandered into this tract, where it is to be hoped they may be preserved most effectually as relics of tribes which soon will occur nowhere else.*

However earnestly one may be urged to witness some of the most striking phenomena, there are many results of Nature's workmanship of less demonstrative character, which the mining engineer ought not to miss. When at Mammoth Springs, do not fail to go up the mountain far back of the terraces with active springs, and, if possible, go down into and explore one or more of the extinct passages. The rapids above the Upper Falls of the Yellowstone are dynamically as interesting as the falling waters, and no engineer should fail to see all the gorge between the upper and lower falls. Approach the upper rapids as the river does—from above. There are few more awe-inspiring sights in the world than to observe the sudden change from dormant to rampant energy as here displayed. In a few moments one can study out the simple but wonderful problem of it all, if one will carefully observe the changes in texture of the rocks in the walls and bed of the stream. No amount of fatigue should induce one to overlook the cliffs in which the buried forest is exposed, near the mouth of the Fork of the Yellowstone. At the geyser basins there is nothing which can be neglected.

It will be impossible for one to do any thorough scientific work in five days, if one sees as widely as possible, but such observations as can be made, if carefully timed, will have much value in connection with others which have been taken; hence, no one should neglect to preserve such gleanings. If no other use can be made of

* Those interested in following up this subject will find it more fully discussed in "The Scientific Value of Yellowstone Park," *Am. Naturalist*, March and April, 1874.

them, the writer will be very glad to classify and publish, with full credit, such as may be sent to him.

The mining engineer, especially in Montana, Idaho, Colorado, Utah, etc., within the volcanic area, has far greater need to understand the phenomena, deposits and subterranean structure of thermal springs than most of our brethren yet appreciate. I know that this opinion is not fully shared by some of the best authorities, but I do firmly hold that the Yellowstone Park area furnishes the key to much of the vein-formation of the West, and I assert most positively that the engineer who does not clearly understand the structure of this tract cannot interpret properly the geology of a large portion of the mining territory west of the 100th meridian.

For a guide to the Park and an interpreter of what our members will see while there, Dr. A. C. Peale's admirable treatise (Hayden's *Survey*, 1878, Part II.) is beyond and above all that has been written.

NOTES ON THE GEOLOGY OF BUTTE, MONTANA.

BY S. F. EMMONS, U. S. GEOLOGICAL SURVEY, WASHINGTON, D. C.

(Utah and Montana Meeting, July, 1887.)

[THE following notes of observations made during a visit of a few days in the month of October, 1886, which was cut short by a sudden summons East, are offered, in spite of their incomplete and tentative character, in the hope that they may aid the members of the Institute, in their coming visit to this region, to a clearer understanding of the general geological conditions under which the ore-bodies occur.]

The Butte mining region lies in a sort of a depression in the mountains which is drained by Silver Bow creek and its various tributaries. On the east it is bounded by high north and south ridges of granite, and on the west by lower ridges, also running north and south, which, as far as examined, were found to consist of rhyolite. High mountains also inclose the depression on the north and south, but at a greater distance than the more immediate bounding ridges on the east and west, the distance between which is hardly more than six miles.

This depression is, in its broader features, like one large valley, but, examined in detail, it is found to be a ramification of shallow valleys, inclosing a network of relatively low, rounded hills, having the topographical form characteristic of an easily disintegrated granite formation. The formation is indeed distinctly granitic, and it is said that no sedimentary rocks are found within fifteen miles of Butte. Yet the mountains which rise so abruptly on the east side of Silver Bow creek are also of granite of the same general variety as the granite of Butte, and the question naturally presents itself, on glancing over the region from some good point of view, Why are these granite hills in the valley so worn down, and those beyond so high and precipitous? It is not that the agents of erosion can have been more active in the one place than in the other; and therefore there must have been some inherent difference in the rocks themselves, that rendered them more susceptible to the disintegrating effects of atmospheric agents. This difference might be found in the fact that the granite around Butte has been subjected to such energetic chemical action from the mineral solutions that have percolated through it, that it has been brought to a state, not necessarily of actual disintegration, but of readiness to disintegrate as soon as it is exposed to the action of atmospheric agencies. This is proved by the waste rock brought out of the mines, which, though not notably soft or easy to drill underground, decomposes and crumbles with comparative rapidity on the mine-dumps. Thus the actual surface-rock would be continually crumbling, and as the disintegrated portions were carried away by erosion, fresh surfaces would be continually exposed to the disintegrating influence of the air.

The main stream of Silver Bow creek flows at first south, along the base of the mountains bounding the valley or depression on the east, and then, just east of the city of Butte, takes a bend to the westward, thence pursuing a general westerly course to the town of Silver Bow, where it cuts through the ridges bounding the valley on the west. At the same time, one of its more important affluents on the north flows first west and then south, through a comparatively open valley, to join it just above the gap at Silver Bow. The two streams thus inclose an area of about three by four miles square, which contains practically all the important mines of the region, and to which my observations have been mainly confined.

About in the center of this area is the "Butte" from which the city of Butte derives its name. This is a very regular conical hill rising several hundred feet above the adjoining ridges, but not as

high as the bounding ridge beyond the valley of Silver Bow creek on the east. It is to the east of a north and south line drawn through this butte, and within the angle of Silver Bow creek, that what were, up to within a comparatively recent period, all the important mines of the district are situated. The hills here are rather higher than to the west of the butte, but have the same general character of softly-rounded slopes. South of them is a considerable open valley, in which Silver Bow creek makes its curve from a southerly to a westerly direction. On the southern slopes of these hills, overlooking this valley, and a little east of the butte, is situated Butte City.

So uniformly are all the rock out-crops weathered and disintegrated, that to one passing hastily over the ground the formation would seem at first glance to be all granite, only differing here and there in the degree to which it has been decomposed. To be sure of having examined all the occurrences of rocks other than granite, or of having distinguished all the minute varieties of this rock, one would require to have made an examination foot by foot of the whole region, and of the underground workings of all the mines in it. In the short time at my disposal I was only able to determine the broad general outlines; and many details have doubtless escaped me.

The rocks I was able to distinguish were granite (two varieties), quartz-porphyry and rhyolite, and of these rocks the granite is the older, the quartz-porphyry and rhyolite being later eruptions.

Granite.—There are two prominently distinguishable varieties of granite: the one extremely basic, the other more than usually acid, to judge by their mineralogical composition. The former I shall call the "Butte granite," as it is the more prevalent variety, underlies Butte City, and includes all great mines which have hitherto given renown to the district. The other, which occupies a more limited area west of the butte, and in which important ore-deposits are of comparatively recent development, I have called, from the most prominent mine within that area, the "Bluebird granite." The Butte granite is a rather dark-colored rock, of moderately coarse but very even grain, consisting of quartz, orthoclase and plagioclase feldspars, and as basic constituents, of which it has an unusually large proportion, of mica, hornblende and augite. This rock is rather remarkable among granites, from the large amount of plagioclase feldspar which it contains, and from the prominence of pyroxene as a basic constituent, much of the hornblende appearing to be only a paramorphic alteration of this mineral. From the specimens of similar

rock brought in from various mining districts in Montana by experts employed under the Tenth census, I judged that it must be tolerably widespread, and a favorite mineral-carrier in that Territory. In my "Geological Sketches" I have called it diorite-granite, from the preponderance of plagioclase feldspar over orthoclase. Its habit is entirely that of granite, however, and an examination of a greater number of specimens of the Butte rock shows that this preponderance is not universal. The Bluebird granite, on the other hand, is a light-colored rock, composed almost exclusively of quartz and orthoclase with a very little biotite. In the extreme type of this rock, as found in cutting foundations for the Bluebird mill, there appears to be only orthoclase feldspar, which is often peculiarly intergrown with quartz, as in the so-called graphic granite, and mica can only be distinguished by the microsope in a few minute flakes. The average rock is fine-grained, showing here and there a small speck of mica visible to the naked eye. It has a sort of granular appearance, almost like a quartzite. In certain transition-specimens a white opaque feldspar can be distinguished from the pinkish orthoclase, which is probably triclinic, and, with the increase in this constituent, the mica increases correspondingly.

In the time available, I was unable to find such evidence of the relative age of these two varieties in contact-phenomena or penetration of one by the other as appeared to me sufficient to definitely determine their relative age. Such as I did find, though it pointed to the later age of the Butte granite, was by no means conclusive. This is a matter of comparatively small importance here, as the ore-deposition is unquestionably later than both granites.

Quartz-porphyry.—This rock, which is apparently of limited extension, was only found by me on Anaconda hill. It is of the normal type, a light-colored rock showing porphyritic crystals of quartz in a felsitic ground mass.

Rhyolite.—This rock presents two varieties of texture, with, however, no essential difference in composition. The rock of the butte and its immediate vicinity is a light-colored fine-grained variety, composed of small crystals of sanidine, quartz and mica in a very subordinate ground-mass. It is of rather massive habit, that is, it shows no regular structure-lines, but it weathers into the thin, angular, sherd-like fragments characteristic of acid volcanic rocks.

The rock from the ridges on the west of the valley, on the other hand, is a typical rhyolite, in that it shows the fine banded structure

so common in this variety of rock and from which the name rhyolite, which signifies "thin-flowing," is derived. Besides this banded structure, in which the bands or layers are only a fraction of an inch thick, the rock shows a distinct bedding, resulting doubtless from successive flows one over the other. This makes it advantageous for quarrying, as it is easily broken out in regular blocks one and a half to two feet thick. In this rock the porphyritic crystals are larger, and at the same time the ground mass is more prominent and shows more glassy or isotropic material than the Butte rock.

Distribution.—In a broad, general way the distribution of the above rocks is as follows:

The Butte granite forms all the country east of a north and south line running through the butte. West of this line and south of the ridge running westward from the divide north of the butte, the Bluebird granite forms the low hills as far south as the valley of Silver Bow creek, and westward nearly to the rhyolite ridge. Butte granite apparently surrounds it, however, both on the south and on the west, that is, it occurs between the Bluebird granite and the rhyolite near Silver Bow, and is found along Silver Bow creek south of the Bluebird mill.

The quartz-porphyry, as far as known, is only found in narrow dikes traversing the Butte granite in the Anaconda mine.

The rhyolite, as found at present, probably occupies a much more limited superficial area than it did originally, having been largely removed by erosion. The butte probably represents the main vent or channel through which it was forced up to the surface. Its outline is irregular and not always easy to trace. On the south of the butte, it extends nearly to a low, rocky shoulder immediately overlooking Silver Bow valley. On the east it extends only to the foot of the steep slope, some distance from the bottom of Missoula creek, which separates the butte from Butte City. On the north it extends to the divide over which passes the upper road running from Butte westward, and is here exposed in a quarry or open cut. Westward from this divide it forms a high ridge, separating the east and west valley of the main northern tributary of Silver Bow creek from those flowing southward into the main stream through the Bluebird granite area. Whether this now connects with western ridges of rhyolite was not observed; but it is likely that the connection may have been carried away by the erosion of the valley to the east of the ridges. A tongue of rhyolite also extends westward into the Bluebird granite area from about opposite the summit of

the butte nearly to the new shaft-house of the Nettie mine. It is probable that there are many other such offshoots extending out into the surrounding granite, both such as show on the surface, and such as can be detected in underground galleries only. One of the latter was observed in the tunnel cut below the Bluebird mill to connect with its engine-room. This is a somewhat irregularly shaped body (as shown in the tunnel, about seventy feet thick), and appears to run near the contact of the acid and basic types of granite. It is to be remarked that springs are liable to occur along such dikes, the reason of which is readily seen to be that the cross-cutting body of later eruptive rock has interrupted old water-channels, and the water must therefore now find its route to the surface along the walls of the cross-cutting body.

Ore-Deposits.—All the important ore-deposits of the region are found in granite; none, so far as my observations and inquiries extended, occur in the rhyolite. On the other hand, dikes of an eruptive rock resembling in some cases rhyolite, in others more like quartz-porphyry, are found traversing the granite in the immediate vicinity of, and parallel to, the deposits. So far as I observed these, however, they were apparently injected previous to the deposition of the ore.

As regards the distribution of these deposits, it is to be remarked that in the Butte granite area, east of the butte, there are two zones of deposits of distinctive mineral constitution, the one being essentially copper-bearing, the other silver-bearing and comparatively free from copper. The former occupies the southern part of the area included between the butte and the bend of Silver Bow creek. In it are the original Butte, Gagnon, Parrott, Anaconda and Mountain View mines, which follow a general east and west line passing through the summit of the butte, and the northern portion of the city of Butte, with many other important mines both on this line and a little to the north or south of it. The principal ores here are bornite, copper-glance and other sulphurets in a siliceous gangue. They contain more or less silver associated with the copper.

North of these are the Alice, Moulton, Lexington and very many other rich silver-bearing mines, in which the ores consist of sulphurets of silver, lead, zinc and iron, and various mixtures of these sulphurets, also in a siliceous gangue but associated with silicate (and probably also carbonate) of manganese, which gives a striking pink color to the gangue-material. At the surface and for some forty feet down, surface oxidation has changed the manganese mine-

rals into black oxides, the silver into chloride and the lead into carbonate. I was informed by a gentleman who has had extensive opportunities for studying the mineral constitution of the various ores of the district, that the manganese minerals are not found in the copper-zones, but are exclusively confined to the silver-bearing deposits.

In the Bluebird granite area, west of the butte, the manganous gossans, or outcrops of the silver-veins, are very prominent on the surface and can be followed for long distances. They consist mainly of oxides of manganese carrying a few ounces of silver per ton, and for a long time were only exploited as a flux for the smelters. Since the owners of the Bluebird mine have developed their deposit to a depth of three to four hundred feet, and have found an almost continuous body of ore, running 40 ounces of silver to the ton and extending many hundred feet in length, an impetus has been given to develop the other mines in depth, prominent among which are the Independent, Burlington and Nettie, which seem to be on the same general east and west line. This, if continued, would run through the butte also. The ores of this portion contain a similar association of minerals to those of the silver-bearing zone east of the butte; but the proportion of metals other than silver is relatively less, and the ore-mass, like the rock which incloses it, is less compact and more easy to work, both in the mine and in the mill.

There are two points of resemblance in, or characteristics common to, all these various deposits, to which I wish particularly to call attention. The first is the direction and character of the fissures along which the ores have been deposited; the second, the manner in which their deposition has taken place.

1. There is a remarkable parallelism in the general direction of all the fissure-planes, both horizontically and vertically; the former being east and west, the latter within a few degrees of the vertical. I say *general* direction; for there are endless local variations from this direction. A given fissure-plane will curve to the right or left, or it will cross transversely at a low angle and merge into an adjoining plane, and again apparently be altogether lost. So the dip, which seems to be prevailingly to the south, may vary, now to the one side, now to the other side of the vertical; and in some mines I was told it would frequently flatten out to a very low angle and then resume its former direction in a short distance. Further than all this, it may be remarked that in a given mine, or series of

mines, supposed to be on the same vein, there are generally several parallel planes along which the ore is found: two, three, or even four in some cases. In those which came under my observation, I did not find, however, that these different planes were very persistent, or preserved any strict parallelism; but that, as I have said before, they might merge into one another, or give out altogether or be taken up by another plane of fracture, a little removed from the strict line of continuity,—in other words, that these so-called veins are rather fissured zones, such as might be produced by some common dynamic movement which would shatter the whole region. Could an accurate map of all the fissure-planes of the region be prepared, I fancy it would show a remarkable resemblance to the diagrams in Daubrée's *Géologie Expérimentale*, which shows the effect of compression and contortion on blocks of various solids, including rocks. To my mind it seems perfectly evident that these fissures are the result of one general dynamic movement. To determine definitely its character, causes and date would require a more extended study; but one naturally connects it in one's mind with that which must have accompanied the rhyolite eruption; and two facts (if such they be) lend probability to this connection. The one is the occurrence of dikes of this rock associated with, and parallel to, the ore-bodies; the other is, that while the principal systems of fissures on either side of the butte would pass through it if continuous, they do not seem to extend up to and be cut off by it, as they would do had they existed previous to this eruption.

It appears, therefore, that these fissures are not like those produced by a single great fault, whose movement might leave considerable open spaces, bounded by two well-defined walls covered by clay gouge and slickensides, to be filled up by foreign mineral matter, as is the commonly received idea of a true fisssure-vein; but that they were simply cracks wide enough to admit the passage of mineral solutions. Hence, where considerable bodies of ore are found, they would theoretically be in a great measure the result of a gradual eating away of the country-rock adjoining those water-passages, and of its replacement by the minerals those waters held in solution.

2. When we come to study the ore-deposits with a view of determining the manner in which they are formed, we find plentiful evidence that it was by replacement of the country-rock rather than by the filling-in of any considerable pre-existing open spaces. In the first place, there is no sharp dividing line between ore and country-rock: the one grades off insensibly into the other. The original fissure-

plane can generally be recognized, it is true; for in most cases the solutions seem to have penetrated the country only on one side of the fissure. Hence the miner often says he has a good wall on one side of his vein and none on the other; but if you examine the country-rock beyond what he considers the limits of his ore-body, you will find it is partially replaced or impregnated with metallic minerals, mostly pyrite. As a rule, the whole zone of rock opened by the workings of a mine is more or less altered, what constitutes pay-ore being arbitrarily determined by the costs of mining and reduction.

The basic constituents of the granite are naturally attacked first, then the feldspars, and finally the quartz itself may be removed, so that in some parts there are found large masses, composed entirely of metallic minerals. It is only in the poorer or transition-portions of the deposits that the process can be satisfactorily traced. Often within the ore-bodies are what look like thin veins of quartz and metallic minerals which have pretty well defined boundaries; these may have been the filling of actual open cracks, but the greater mass of the ore-bodies is undoubtedly a more or less complete replacement of the granite country.

Attention has already been called to this fact by that keen observer, Mr. Richard Pearce,* who as early as 1864 had traced a similar process of replacement in the tin-deposits of Cornwall. In an extended description of some of these deposits, published in the *Quarterly Journal of the Geological Society* [1878, p. 640] by Mr. Pearce's successor, Mr. C. Le Neve Foster, many striking points of resemblance with those at Butte may be found. Not only are the ore-bodies a more or less complete replacement of the granite country-rock, but they seem to follow fissured zones, similar in some cases to those described above, in others constituting an irregular network or stockwork, as Mr. Foster calls it.

The first practical deduction I would make from this condition of things is that, after all, the particular form in which ore-bodies occur is a matter of little importance, as regards their respective value, and that the prejudice in the mind of the average western miner in favor of what he calls a "true fissure-vein" is without a valid foundation from a geological standpoint. It also shows the inadvisability of having mining laws so framed that the title to a mining

* *Jour. of Roy. Inst. of Cornwall*, No. XXVI., 1883.

claim may be dependent upon the definition, current at a given time, of a geological term. The owner of a mine, which contains some of the most magnificent ore-bodies in this remarkable district, told me that he had been unable to obtain a U. S. patent for his claim because he could not show a well-defined wall to his vein. It is easy to see, moreover, that under the present mining laws such a condition of things is bound to give rise to endless litigation, when the ore-bodies commence to cross the side-lines of claims in depth.

As regards the source of the metallic minerals, nothing more than a surmise could be formed in the short time that I was in the district. I should naturally look for this source in some of the bodies of eruptive rock that occur here, but not necessarily in such parts of them as are found at the surface. For my view of the theory of lateral secretion, as it is generally called, is, not that the ore was derived necessarily from the immediate wall-rock of the deposit, but that it may have been brought in from a body of rock at some considerable distance therefrom, provided conditions were such as to admit reasonably free passage of the mineral-bearing waters from it to the locus of the deposit. Here it seems probable that the rhyolite may have a wider extent below than it does at the surface, judging from the number of different points at which it penetrates the granite, which number would probably be largely increased by a more detailed examination. The heat of such an eruptive body, cooling, as it must have done, in depth, at an extremely slow rate, would have rendered much more energetic the action of percolating waters and would have facilitated their circulation. Although it is not now possible to reach these underground bodies, an examination of the comparatively unaltered surface-rocks, if made on a sufficiently extended scale, would afford means of determining the probable source. I had the freshest among the specimens I collected carefully tested for silver by crucible-assay, using a large amount of material, and there was found:

In the rhyolite, 6–100 oz. Ag per ton,
In the Butte granite, 9–100 oz. Ag per ton,
In the Bluebird granite, none.

Inadequate though such single tests are, they show a fair possibility that either the granite, or the rhyolite, or both, might have contributed the material to form the ore-bodies.

Discussion.

W. A. Clark, Butte City, Montana: It is true, as Mr. Emmons says, that in this district the miner often "has a good wall on one side of his vein and none on the other;" but many veins here, and particularly large ones, have two distinct walls. I may instance the Elmer Lee and the Gambetta.

J. A. Legget, Butte City, Montana: To the mines mentioned by Mr. Clark I might add the Champion and the Gold Flint, which have well-defined walls, though the veins are narrow.

George W. Irvine, Butte City, Montana: Doubtless in some instances the requirement of our local law, that a vein when located shall have at least one defined wall, has led to the neglect on the miner's part to explore carefully for the other wall, when one has been developed.

R. W. Raymond, New York City: The occurrence of one distinct and one indistinct wall is frequent among mineral veins, and constitutes one of the alternative results of certain simple causes; and these causes are equally admitted by the advocates of the ascension and those of the lateral-secretion theory of vein-formation. Indeed, as now stated, these theories come so near together as to leave little ground for dispute between them. When the ascensionist declares, as Cotta did, that his ascending mineral solutions are derived from the country-rock in depth, and the lateral-secretionist explains, as Mr. Emmons does, that his lateral secretion may have taken place in the country-rock below, it is hard to see how they differ, except that the latter presents a more general and elastic formula, capable of including the case of ore-deposits actually formed from the rock adjacent or near, on the same level, or even a higher level. It is a question, not of theory, but of analysis and observation, what has been in each case the source of the mineral solutions. Attempts to demonstrate this source to be the country-rock, as seen at the surface or in the mines, have failed in many cases, succeeded in a few cases, and in most cases left the matter undecided. In the latter category I would class the results of the analyses of Butte rocks given by Mr. Emmons.

The existence of one or of two defined walls may be due, in the first place, under either theory of infiltration, to structural or other differences in the walls themselves. When the deposit occupies a "contact," the rocks of the hanging- and those of the foot-wall may differ in metalliferous character, in susceptibility to decomposi-

tion and metamorphosis, and in conductivity for heat; and these differences may affect not only the distribution of ore and gangue, but also the relative disintegration and obliteration of the two original boundaries of the fissure through which the mineral solutions entered.

Even in the case of deposits occurring between walls of originally similar rock, it must be remembered that movements of the walls may produce physically an unequal comminution or disintegration, or may shatter and fissure one wall more than the other, or, finally, when extensive, may bring opposite to each other as walls portions of the original mass of country-rock which are, by reason of the different depths at which they were cooled or crystallized, etc., physically unlike—as, for instance, the deeper parts of an eruptive mass, as compared with the upper parts, or the coarser conglomerates as compared with finer-grained sandstones of the same series.

Setting aside these causes of actual distinction between the two walls, we may consider the simple case of a fissure opened between two similar walls and traversed by mineral solutions.

If these solutions come from below, and not from the country-rock on the same level, and if they do not react chemically upon the country-rock, and if the fissure has a steep dip, the probable result is a banded structure, such as the old-fashioned typical fissure-veins show—often accompanied with bi-lateral symmetry in the arrangement of the successive mineral layers, like the symmetry observable in geodes of amethyst and agate.

If the solutions, however, react upon the rock, decomposition and interchange may take place to such an extent as to destroy both defined walls and nearly or quite obliterate the original fissure. It is a notable fact that this has occurred in many tin deposits—a class once referred to plutonic agency. Because the tin-ore is found impregnating eruptive granite without definite fissures it was supposed that it must have been erupted with the granite.

Again, if the solutions came from the country-rock, at or above the level of the ore-deposit, it is most likely that the hanging-wall will be more or less obliterated, and the foot-wall relatively distinct. This follows from two considerations: first, because the rock of the hanging-wall is in such a case the main source of the water-supply for the fissure, the natural effect of gravity (apart from special water-channels) being to carry the waters of the foot-wall away from the vein, and to some other vein below it; and secondly, because the disintegrated rock of the hanging-wall would naturally deposit

débris on the foot-wall, protecting it from disintegration, and thus preserving its definiteness.

An interesting example of such action is furnished by the salt-mines of the Salzkammergut, in the Bavarian Tyrol, where the salt is obtained by *Kammerbau*. Large chambers are excavated in the mass of the mountain of saliferous rock; these are filled with water, and after the water has become saturated with salt from the rock, the brine is drawn off for evaporation. It is found that these chambers, instead of growing larger in all directions through the action of the water, advance upward through the mountain, because the earthy material, falling from the roof as its salt is dissolved, protects the floor. In a similar way, a fissure-vein under the conditions assumed, might grow towards the hanging-wall side until the process itself had choked the channels of the water-supply. The result would be "a good foot-wall, and no hanging." On the hanging-wall side the ore-deposition would simply grow more and more scanty as decomposition and interchange had grown feebler, and the boundary would be a commercial one—the line, namely, at which the ore ceased to "pay."

It is my impression that in the great majority of instances, where one wall is better defined than the other, it is the foot-wall. It may be inquired, however, whether any simple cause can be suggested for the opposite (and, I think, far less frequent) phenomenon of a defined hanging-wall, with an indistinct foot-wall. This is more difficult to account for; and in my judgment the cause is to be sought in special local conditions. It may be remarked, however, that decomposition unaccompanied by replacement, as for instance, the simple solution of carbonates in water containing carbonic acid, might naturally, by mere force of gravity, operate upon the foot-wall rather than the hanging. This is, however, likely to be confined to the surface-zone of a deposit, and to play little part in the real problem.

My own observation underground has been that a good hanging-wall on a fissure-vein is usually *very* good; that it is likely to show slickensides, and to carry a protecting selvage or fluccan. This suggests at once a cause which may counteract the effect of decomposing solutions—namely, the formation, through movements of the rock, or through infiltration of mechanical sediments, of a protective layer on either wall.

Finally, I would suggest that the case of *wide* veins with two well-defined walls, alluded to by Mr. Clark, is a somewhat different

one. In that case, I suspect, we generally have two parallel fissures and a zone of broken ground between, laid open by its shattered condition to the action of metamorphosing and impregnating solutions, and thus gradually becoming "vein-matter." The bounding fissures need not be the channels of the ore-supply. They may (like the successive "east walls" of the Comstock) be in themselves insignificant, *except as they are filled with clay*, and thus constitute water-tight boundaries. The amount of water which one inch of clay will hold back is incredibly large—to anybody but a miner.

This subject is both important and interesting, and cannot be too much discussed. What we need, however, is more and closer observation of facts, to give us the basis for safe induction. Prof. Comstock, in his paper on the Yellowstone Park, read at this meeting, has urged that a study of the volcanic phenomena there exhibited is directly contributory to a knowledge of the vein-formation of this whole region. In that I agree with him. We cannot afford to neglect any phenomena or analogies by which the subject may be illuminated.

THE ASSOCIATION OF MINERALS IN THE GAGNON VEIN, BUTTE CITY, MONTANA.

BY RICHARD PEARCE, DENVER, COLORADO.

(Utah and Montana Meeting, July, 1887).

WHILST most of the silver- and copper-bearing veins of Butte have characters somewhat similar, the Gagnon vein has certain rather remarkable features which are not noticed in any of the other mines.

The silver-veins proper, as illustrated in the Alice, Lexington, Moulton, and many others, have certain well-known typical characters. Manganese is always present in variable quantities as rhodocrosite or rhodonite, associated with quartz, pyrite, and sphalerite, and with little or no copper.

The copper-veins also have certain mineralogical characteristics, which are very similar, or different only in minor points. Chalcocite, chalcopyrite, bornite, enargite, with variable quantities of pyrite, may be said to represent the general mineralogical features of these lodes. It is worth noticing that while the silver-bearing lodes have more or less sphalerite, this mineral is almost entirely

absent from the copper-mines; and the same may be said of manganese minerals.

The Gagnon stands alone in this respect, and shows an association of minerals which is unlike anything else to be seen in the district. Perhaps by far the most interesting feature is the occurrence in very large quantities of a rich silver- and copper-bearing zinc-blende. This mineral was discovered between the 200- and 300-feet levels of the Gagnon, and some thousands of tons of it were mined and smelted, principally at Argo, Colorado. Smelter-returns show varying contents of shipments as follows: silver, from 50 to 200 ounces per ton; copper, from 1 to 42 per cent.; and zinc, from 7 to 48½ per cent.

The ore may be described as a brownish-black mineral, with a small amount of quartz and barite, but without any indications whatever of the ordinary copper-bearing minerals. It was perfectly homogeneous, and the only guide one had in estimating the amount of copper it contained, was the slight variation in color from brown to black, the blacker varieties being richer in copper. I have every reason to believe, judging from the peculiar structure of the mineral, and the almost entire absence of crystallization, that it is a massive wurtzite, which is zinc-blende occurring in a different crystalline system, viz., the rhombohedral. After some years of careful searching for a crystal of the mineral, I was fortunate enough to discover some small crystals (having the form of the hexagonal prism, with the planes of the pyramid), which are undoubtedly wurtzite.

This occurrence of zinc-blende, containing so large a percentage of copper, is indeed remarkable. I think it probable, however, that the varieties which are so high in copper have undergone some change, the copper being enriched by the oxidation and partial disappearance of the zinc. Very fine specimens of goslarite (sulphate of zinc) are now to be found on the walls of the levels in the Gagnon, which would indicate beyond a doubt that the zinc-blende has been oxidized to a large extent. I have also reason to believe that the same influences which effected the oxidation and removal of the zinc may have caused the disappearance of a portion of the silver, for we find as a rule, in the lots of ore smelted, that an increase in the percentage of the copper accompanies a corresponding decrease in the silver-contents, and the presence of native silver in very thin plates throughout the masses of the blende would indicate some sort of secondary deposition of the silver.

In the deeper levels of the mine, pyrite is more abundant than in the upper workings; and there is a marked decrease in the quantity of zinc-blende.

Scattered through the lode we often find a bronze-looking pyrite, having a fracture somewhat different from that of ordinary pyrite, namely, conchoidal; and this mineral is often rich in silver. Some specimens which I have examined contained as much as 3 per cent.

A mineral which was found about a year ago in the 500-feet level in very small quantities interested me very much, on account of its richness in silver, and its peculiar composition. An analysis of a selected specimen gave:

	Per cent.
Copper,	41.10
Silver,	24.66
Zinc,	9.80
Iron,	2.09
Sulphur,	20.51
Insoluble residue,	1.02
	99.18

I have been inclined to the belief that this is a new compound (a copper-silver-zinc mineral).

If we regard the iron as replacing a portion of the zinc, we arrive at the formula:

$$3\,Cu_2S + Ag_2S + ZnS.$$

Whether the zinc may be regarded as an accidental admixture or as part and parcel of the mineral, I am not prepared to say; but judging from the uniform character of the mineral, I should be inclined to believe that zinc enters into its composition.

Enargite has been found rather sparingly in the Gagnon vein; but recently a specimen of this mineral was found in the western part of the mine, which was remarkable for its richness in silver. As a general rule, enargite is poor in silver; but this particular specimen to which I refer had all the properties of enargite, only that it contained about 10 per cent of silver.

The only manganese mineral I have ever found in the Gagnon, and that only in one or two cases, is hübnerite (tungstate of manganese). This is rather remarkable, when we consider how uniformly manganese minerals are distributed in nearly all the veins which surround the Gagnon.

Bornite occurs sparingly, and is generally rich in silver. Chalcopyrite is also found, but it is usually low in silver.

The gangue-matter of the vein is quartz and feldspar, with an occasional scattering of barite.

THE RAINBOW LODE, BUTTE CITY, MONTANA.

BY WILLIAM P. BLAKE, NEW HAVEN, CONN.

(Utah and Montana Meeting, July, 1887.)

THE Rainbow Lode is situated at Walkerville, in the Summit Valley mining district, Silver Bow County, Montana Territory, about one mile from Butte City. It was so named in 1876 by Mr. J. E. Clayton, in recognition of the broadly-sweeping curve of the outcrop across the rounded surface of the eastern slope of a granitic mountain, at an elevation of about 6000 feet above the sea-level. It is on the western side of the range which divides the waters of the Missouri from those of the Columbia.

History.—The history of the district is an instructive one. The water-courses and flanks of the hills were worked for years for placer gold. Millions of dollars' worth of low-grade gold have been washed out of the surface gravels below the veins of the district. Missoula Gulch and Silver Bow creek, just below Butte, are noted for their rich placers, and they continue to yield largely, producing within narrow limits from $50,000 to $100,000 dollars' worth of gold-dust yearly. It is ragged vein-gold, and evidently is near its parent source. It is no doubt derived from the disintegration of the veins of the district, and this conclusion was soon forced upon the miners. The veins which were seen outcropping boldly upon the slopes above them were located, and prospecting commenced. These outcrops were found heavily stained and blackened with oxide of manganese, and were porous and honey-combed in places, showing that the quartz was formerly filled with pyritous ores, and that these ores had decomposed and dissolved away, leaving, however, portions of the precious metal behind, especially at and near the surface of the ground. Assays of the croppings gave variable and often contradictory results; but the fact that these were the outcrops of lodes bearing gold and silver in depth was patent to any experienced observer of mineral veins.

The lodes of the district were first brought prominently to notice by Mr. Clayton in 1876, and in October of that year the Alice mine, upon the central portion of the lode, was opened by Mr. Marcus Daly, now so well known as the superintendent of the Anaconda Copper mine, upon another extremely interesting lode, in the same region.

The mining developments upon the Rainbow lode have since been carried on without cessation. Many millions of dollars have been expended and returned, and cities and villages cover the surface which, ten years ago, was almost a wilderness. We owe to the mines of Butte not only this remarkable local development of population and wealth, but a great stimulus to industrial development elsewhere, and, also, the construction of the line of railway which permits us to ride in luxurious palace-cars almost to the mouths of the shafts. But it is not the object of this paper to give a general history of the mines of Butte. The Rainbow lode is selected as that one of the best known and typical examples, with which, by reason of several careful examinations, I am most familiar. One of my visits of inspection was made in company with our fellow-member, Professor George W. Maynard, and our joint reports upon several of the claims upon the lode are in print. The lode is now chiefly owned and worked by the Alice Gold and Silver Mining Company and the Moulton Mining Company. The opening of these mines is due chiefly to the courage and enterprise of the Messrs. Walker Bros., of Salt Lake City, and to the Messrs. Clark of Butte. The Alice, for several years past, has been under the superintendence of W. E. Hall.

The lode is opened by nearly continuous workings below ground for more than a mile in length, and it has been pierced at intervals by shafts, the principal one at the Alice being 1000 feet deep, and that at the Magna Charta, 700 feet.

The claims located upon the lode, beginning at the Rising Star on the west, and in succession northeastwardly are:

	Feet.
Rising Star,	1080
Frank Moulton,	1466
Alice,	1183
Fraction,	218
Magna Charta,	1500
Valdemere,	300

These, with the exception of the Moulton and the Fraction, are owned and worked by the Alice Company.

The Country-Rock of the Lode.—The Rainbow lode is well defined and clearly marked by massive siliceous croppings, metal-stained and honey-combed. These croppings are visible from a distance, and can be followed by the eye in a general northeasterly and southwesterly direction, across the undulating surface of Summit Valley. Close examination shows them to consist of layers and parallel masses, differing somewhat in composition and in hardness.

The softer layers do not crop strongly, while the harder and more compact quartz stands boldly out to a height of ten to twenty feet more. These croppings can be followed for the whole length of the claim and beyond it to the east and west. The Alice claim includes the most massive and well-defined portion of this line of outcrop.

The country-rock is a well-defined gray granite, of even and close grain, with a large proportion of black mica in small crystals, but little quartz and, in some places a little hornblende, becoming syenitic. It has no well-defined rift or slaty structure, being apparently eruptive or plutonic in origin, rather than metamorphic. It is much more friable and soft than the rocks of the adjoining range, forming the continental divide.

Although the granite appears to be a homogeneous, crystalline, and plutonic rock, there are some reasons for regarding it as possibly an altered rock, originally of mechanical origin.

The celebrated conical hill, known as "The Butte," which gave the name to the place, has generally been called an outburst of porphyry or trachyte. It is generally regarded as having broken through the granite, and as probably having some close connection with the origin of the mineral veins which pass beyond and around it, but do not traverse its substance.

In going south and east from the Rainbow lode, around the head of Missoula Gulch, the granite becomes less crystalline and homogeneous, and is more granular and quartzose in its structure and composition. In some places it appears to pass into, or to inclose, large masses of quartzite. The Butte is decidedly a granular rock; apparently an altered sand-rock. It is full of irregular grains of quartz, in an amorphous, felsitic matrix or paste, also containing small crystals of biotite mica. There are even traces of the ancient stratification, with a north and south trend, but the rock cannot be traced far in the line of strike. The Butte is an isolated, island-like eminence surrounded by granite.

The granite directly back or west of the Butte is in part hornblendic, and, still further west, becomes a coarse aggregate of quartz, feldspar, and biotite; the same constituents are found at the Butte, but entirely different in appearance.

At the Magna Charta claim, the dark-colored hard granite is micaceous and not hornblendic; but some hard outcrops, southwest of the Alice shaft, contain hornblende as well as mica. Small crystals of sphene are common, and minute grains of pyrite are disseminated in the mass of the rock.

Throughout this granitic area there are alternating belts or spaces

decompose faster than in the
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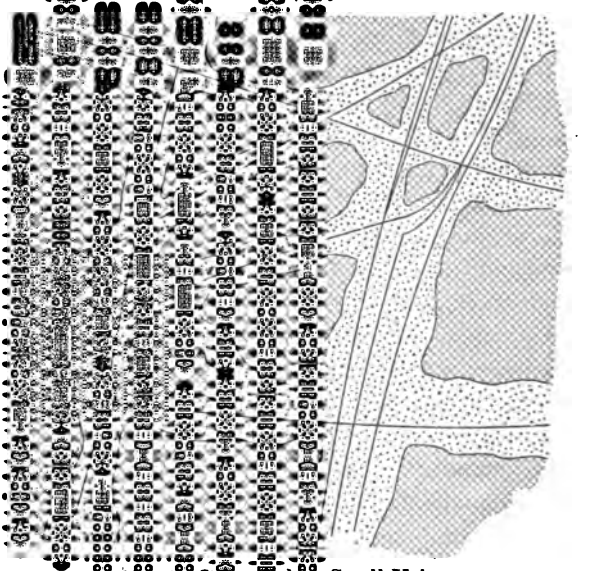

Small Veins.

pheric degradations, preceded,

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of the shaft, where a chamber
anite is traversed in every di-
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to a yard or more in diameter.
ks are hard, dark-colored, and
cted by the veins, while the

rock next to the veins and seams is softened and partially kaolinized, turning a light gray color. The veins seem to have affected the rock on both sides. These narrow veins extend for many feet with an even width, and do not fault or throw each other at the places of intersection. The vertical veins, from one to six inches in thickness, are filled with sulphides, while the highly inclined or horizontal seams are nearly barren. These facts, while interesting to the student of vein-formation, have also a practical bearing, being indicative of an extensive fracturing or, at least, a deep-seated mineralization of the granite.

The Lode.—The general direction of the lode is northeast and southwest, magnetic.* The dip varies, being generally westward, but at a high angle. At the old Alice claim the dip at the croppings, and in the upper levels, is from 17° to 19° from the vertical, or 71° to 73°. At the Magna Charta, the dip is also westward, diminishing as depth is attained, as may be noted by inspection of the section, Fig. 3.

At the Rising Star, where there has been extensive local disturbance, the dip for a short distance is reversed. A part of the lode is here turned over.

There are many local dislocations or side-throws of the lode, visible upon the surface in tracing the croppings. They may be seen at the south end of the Moulton ground, and on the Rising Star location. They are also found upon the Fraction, and on the Magna Charta, especially near the side-line between this claim and the Moose.

In going south and west along the lode, it is seen to split up into several branches, which seem to diverge one from the other like the fingers of a hand. We thus find numerous parallel locations covering these branches; amongst them, the Silver Safe, Goldsmith, and the Consolidated properties. At the other end of the lode, north and east, we have the Valdemere, Poser, Boston, and others, all showing ores similar in their nature, but separated by considerable masses of granite.

The lode, taken as a whole, may be described as a compound one, being in some places divided up into two, three, or more contiguous portions, separated by granitic rock in large lenticular bodies, or "horses." In the Alice claim the granite of the hanging-wall is firmer and broader than the underlying granite. It is less decomposed, and appears to be the limit of the lode on that side, while on

* The deviation of the needle at Walkerville and Butte is about 21° East.

the eastern, or lower side, the division between the lode and the rock is less certainly defined. On the hanging-wall side the quartz vein-stone is generally firmly adherent to the granite, being "frozen on," as the miners say, while on the opposite, or eastern, side the quartz vein-stone is separated from the country by a succession of clay seams and broken ground. The lode, in fact, is the plane of breaking and movement; the movement, in places, being along the plane of contact between the granite and the vein-stone, and, in other places, in the midst of the vein-stone. Such planes of movement are marked by the usual fracturing and comminution of the lode and the formation of clay selvages. Portions of the vein are brecciated. In some parts of the lode the separation between the vein-stone and the rock is marked by hard, polished walls, almost mirror-like in smoothness, but striated in one or more directions according to the movement, or sliding. The phenomena go to show, in general, that there has been considerable lateral movement, and that this has taken place chiefly along and near the eastern part of the vein; while the western part, being more firmly united to the granite, has held its position, although it has been broken and abraded near the line of movement. The ore-masses in the body of the lode are lenticular in form and lie side by side. Where one thins down, another increases in thickness and takes its place, or is replaced by one of the granitic horses. Some portions of the vein-mass are thoroughly crushed and broken up; as completely so as if they had been passed through a crusher, while the fragments are partially rounded by attrition one upon the other. On level III. of the Alice there is a mass of this kind, 29 feet in thickness, visible in the cross-cuts from the shaft.

A few sections will illustrate the structure and thickness of the lode at different depths.

On level I. of the Alice, 100 feet from the surface, the main shaft descends through the center of the lode. The total thickness is about sixty feet. The section is as follows:

Cross-Section of Lode on Level I. of the Alice Mine.

	Feet.
i. Granite of southeast wall,	...
ii. Clay wall,	...
iii. Quartz vein-stuff shattered and broken,	27
iv. Quartz " first-class milling ore,	12
v. Quartz " black with oxide manganese (poor),	12
vi. Quartz " "hard ledge,"	9
vii. Granite, northwest wall, country-rock.	...
	60

All this portion of the vein is rusted and brown in color, the pyrites having been oxidized and washed out, leaving, however, the greater portion of the precious metals behind in a condition to be easily amalgamated without roasting.

On level II., 109 feet lower, the ores are but little oxidized and are in their normal condition, being below the reach of atmospheric water and oxidizing agencies. The succession of quartz-bodies in the cross-cut from the shaft is as follows:

Cross-Section of the Lode on Level II.

	Feet.
i. Clay wall mixed with quartz,	2
ii. Black pyritous ore of high grade,	1
iii. Granite horse,	2
iv. Broken and shattered quartz, low-grade ore with seams and nodular masses of manganese spar,	30
v. Clay wall,	...
vi. Coarse broken quartz, second and third class ore,	12
vii. Clay wall,	...
viii. Hard quartz ore with veins of manganese spar, first and second class ore, good in stopes above,	28
ix. Granite of northwest wall, country-rock,	...
	75

On the third level, where the large body of crushed ore occurs, the vein is 80 feet in thickness and is made up of several ore-bodies, presenting a very interesting section from wall to wall.

Cross-Section of the Lode on Level III.

	Feet.
i. Granite of southeast wall at shaft, with heavy clay wall mixed with quartz,	...
ii. Flinty quartz,	11
iii. Crushed vein-stone, rounded and irregular fragments,	29
Clay seam,	...
iv. Second-class ore with granite horse,	8
v. Black clay wall, well defined,	...
vi. Milling ore (drift), taken out in parts,	9½
vii. Hard quartz,	8
viii. Hard quartz ore with veins of manganese spar,	12
ix. Hard country granite with veinlets of quartz and of manganese spar,	...
	77½

At the 700-foot level, the lode still shows a large amount of softening and alteration of the granite on the foot-wall side. There is, also, some softening and change on the hanging wall, as shown

level, representing an aggre-
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36-foot vein, of ore resem-
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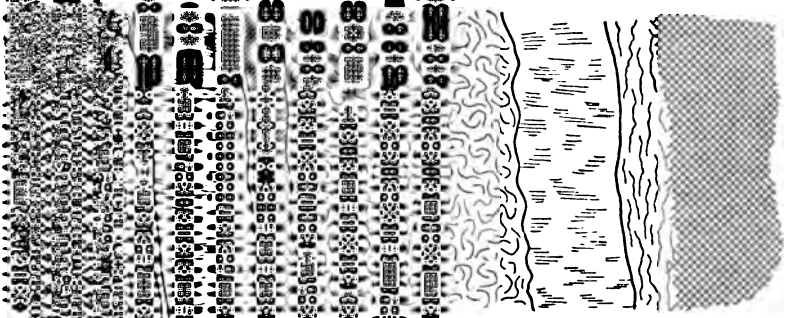

7 8 9

the 700-foot Level.

"curly ore."

hanging-wall country.

crosscut exposes the veins
king four branches, or four
ickness of each of these veins

	Thickness.
	39
	6½
	10
	6

thickness of 61 feet 6 inches,

[illegible]-foot level, ex-
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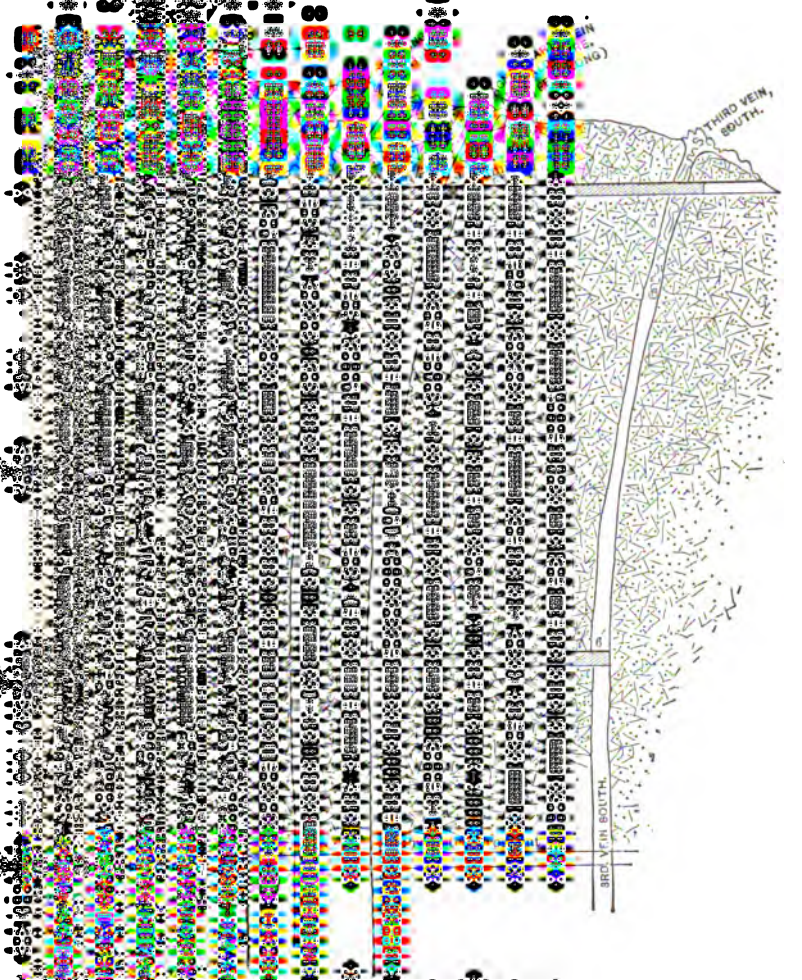
[illegible] veins are best
[illegible]



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[illegible]hesses, approxi-

Mineralization of the Lode.—The general nature of the ore has already been foreshadowed in the description of the vein. It may technically be divided into two chief classes: the oxidized and free-milling, and the pyritic or sulphurous, which requires artificial oxidation or roasting. The first class is found from the surface to a depth of 150 to 200 feet; some of the oxidized portions extend as far down as the 200-foot level. Below that the quartz is white, and the metalliferous contents are bright and glancing, giving a brilliant metallic ore, carrying iron pyrites, galena, zinc blende and sulphide of silver. All these minerals are disseminated in the quartz in a granular, sub-crystalline condition, and in general are in close mixture and association, the exception being the iron pyrites, which is frequently disseminated in the gangue, a little apart from the galena and blende.

An analysis of this ore in 1878, showed the presence of 30 per cent. of silica, about 14 per cent. of lead, 15 per cent. of iron, and 5 per cent. of zinc, with 21 per cent. of sulphur, and silver and gold to the value of $57.50. The percentage of silica or quartz in the ore since worked is higher. Two trials, one upon the average dry crushed ore from the batteries and the other upon the pulp after roasting, gave me

	Raw Ore.	Roasted Ore.
Silica, ...	60	65 per cent.
Soluble matters, ...	40	35 "
	100	100 "

Fluor spar of a beautifully light green color, the chlorophane variety, occurs in the upper level of the Rising Star claim. Galenite is frequently disseminated in the massive manganese spar in small grains and crystals.

Wire-silver occurs not only in the quartz vein-stone, or with the well defined ore, but sometimes in the granitic rock of the horses or divisions of the lode, apparently disconnected with any vein-stone.

Pink manganese spar—the species known as rhodochrosite—is the most characteristic mineral of the vein-stone of the lode. It occurs in large quantity in the massive form, and occasionally in good rhombohedral crystals in vugs or cavities. The massive variety is frequently thrown out in large blocks 12 to 18 inches in diameter, with a bright rose-color when moist and recently mined, which becomes altered on exposure to the air and sunlight. It is generally much intermingled with quartz, which spreads through the substance of the spar to such an extent as to partly disguise it,

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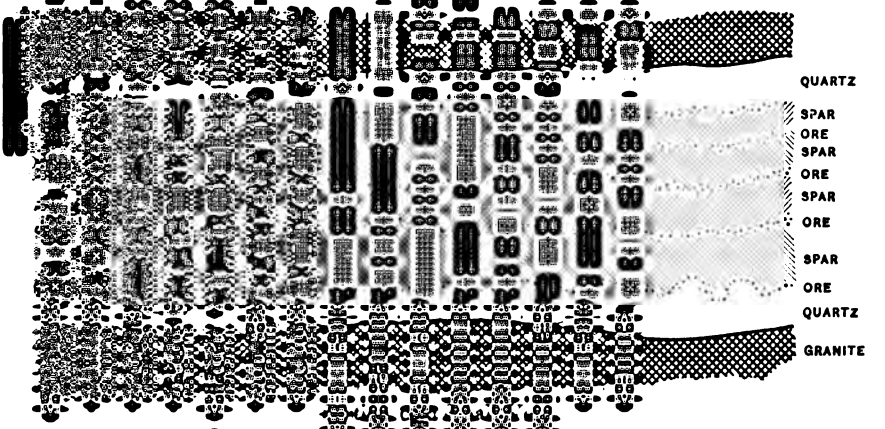

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levels of the mines. Masses of the "blue ores" are often covered with such filaments and wires of silver, so thickly matted and curled over the surface, as to resemble a woolly covering. These filaments are often several inches in length. They are, in general, fine, and not much thicker than horse hair or jewellers' binding-wire; but specimens have been taken out which exhibit the silver in curled masses, one-eighth of an inch thick, forming beautiful cabinet specimens. There are, in addition to this filamentous form of the silver, flake-like scales and patches of the metal, spread upon the surface of rich rock, and also impregnations of the metal in a thicker and less crystalline form through the body of the quartz, and especially in the median line of the veinlets of quartz, filling, together with silver sulphide, the spaces between the small interlacing crystals of quartz. It is this disseminated silver which, no doubt, adds greatly to the bullion product.

Gold is also an element of importance in this ore. It adds considerably to the value of the bullion, and justifies naming the ore a *gold-* as well as silver-ore.

That this metal would be found in the vein, there could scarcely be a doubt, when the nature of the placer deposits just below it was considered. Its presence cannot be detected by the eye in the undecomposed ores, and it has rarely been seen in the oxidized ores of the upper levels. Its presence, in connection with the silver throughout the Alice mine, is shown by the assays, and it is obtained in working. The percentage or yield of gold, relatively to the silver, was notably greater in working the free-milling ores, than it has been in treating the pyritic ores below. The effect of the natural oxidation has been to liberate the gold as in ordinary auriferous veins, and it is readily collected by the process of amalgamation.

The free-milling ores of the upper levels show but little metallic-looking ore. There are some stains of oxide and phosphate of lead, at and near the surface, and the highest grade of ore below is studded with slender silky crystals of carbonate of lead.

The assays from the battery-averages of ore crushed in bulk from the free-milling ore of the upper levels ran high, generally over one hundred dollars in value, twelve to thirteen dollars of which was in gold. The bullion, up to the time of commencing to roast the ores, contained, in value, 87.26 per cent. of silver, and 12.74 per cent. of gold. After the treatment of the ores by roasting commenced, the bullion contained, per cent.:

Silver,	92.93
Gold,	7.07

The assays of the tailings account for this difference, showing a very considerable loss of gold as compared with the loss on the silver.

The gold is alloyed with a part of the silver. That is, a portion of the silver contains the gold, while the filamentous silver and the sulphide are without it.

An experiment with a small quantity of silver-like metal, obtained from specimens of the ore, avoiding the filiform silver, tends to confirm this opinion. On dissolving the silver by the usual method followed in quartation, a residue of gold was obtained. This indicates the presence of an alloy of the two metals resembling the *electrum* of Pliny and the ancients. In all the assay-tests of the ore, the gold is largest in quantity where the silver is most abundant. It is not found where silver is absent or in small quantity. Tests made of the pyritous ore, chiefly iron pyrites, gave only a trace of gold.

There are some good reasons to support the opinion, that the silver and gold is not disseminated in the bulk of the ore, but is found chiefly in certain layers or portions of it, and in a comparatively free state, not enveloped or combined with the baser metals or their sulphides. This suggests the possibility of working the ores without roasting. Some of the massive sulphides of lead and zinc, almost free of quartz, contain but little silver. Other portions are extremely rich. The quality of the ore cannot be safely judged by the eye.

The 4425 tons worked raw at the Alice Mills, to July 23, 1878, produced 122 bullion-bars valued at $328,417.43, averaging $74.22 per ton worked.

The 8941 tons of ore worked, roasted, produced 302 bars, valued at $509,121.96, or an average of $56.95 per ton worked. The total value of the 424 bars of bullion was $837,539.39.

The average weight of the bars for that period was 1880 ounces. The average fineness of bars produced from the raw ore was 985, and from the roasted ore 725.

The bullion shipped during the two periods of working the raw and the roasted ores contained silver and gold in the following amounts, by value.

Value of Silver and Gold in the Bullion from Raw and Roasted Ores.

Bullion from Raw Ore:

			Per cent.
$328,417.93	Silver,	$286,572.23	= 87.26
	Gold,	41,845.20	= 12.74

ed Ore:

	Per cent.
. .	$473,147.32 = 92.93
. .	35,974.64 = 7.07

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uch in value, that the average
has not exceeded twenty-nine
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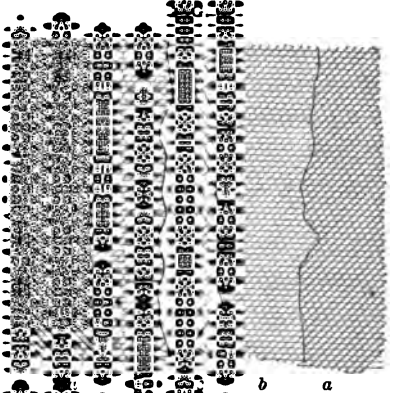

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d by Decomposed Granite.
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at Butte, Montana."

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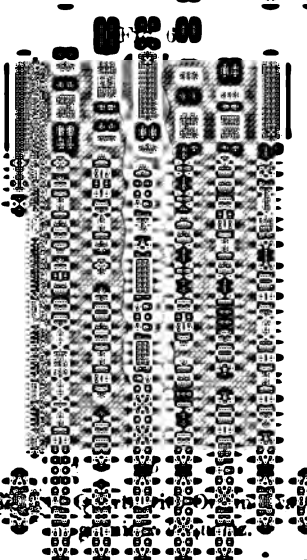

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with a central filling of manganese spar and ore. The alteration of the adjoining granite is interesting and significant. The phenomena favor the view of the origin of the vein and ore from lateral secretion, or rather from the slow flowing of the water contained in the mass of the rock towards a seam, fissure, or crack, along which the solutions could flow with less resistance than in the capillary pores of the rock. It would seem that the quartz of the veins is derived from the adjoining granite, for we find most of the quartz removed and even the silicates decomposed, with a loss of at least a part of their silica.

So, also, the sulphides may be derived from the mass of the adjoining rock, and become concentrated along the plane of the flow in the fissures.

It is important to note, in this connection, that, at the deepest point yet reached on the lode, 1000 feet below the surface, and far below the permanent water level of the country, the amount of decomposition of the granite does not appear to be less than above. The alteration of the rock does not appear to be dependent upon atmospheric or surface agencies. One cannot fail to be impressed also with the idea that the operations of vein-forming are still going on at the ordinary temperature of the mine. The permeation of the granite by hot water and steam does not appear to be required for the alteration of the rock; yet it cannot be affirmed that these agencies have not acted at some former period.

FURTHER NOTES ON THE HYDROMETALLURGY OF COPPER.

BY T. STERRY HUNT, MONTREAL, CANADA.

(Utah and Montana Meeting, July, 1887.)

In a paper presented to the Institute in 1881* I have discussed at some length the question of the "Hydrometallurgy of Copper," and among others, have noticed two methods, devised and patented by Mr. James Douglas and myself, for the extraction of copper

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* *Trans.*, vol. x., pp. 11–25.

from its ores. Of these, the first is based on the action of dissolved ferrous chloride on the oxide of copper, and the second on the employment for its solution of the dilute sulphuric acid got by the action of sulphurous acid gas on a solution of sulphate of copper containing only so much chloride as is required to convert the copper into cuprous chloride. This last method was devised with special reference to the separation of pure copper from silver-bearing ores.

In the first of these methods, patented in 1869, the ferrous chloride is dissolved in strong brine, and by its action with cupric oxide gives rise to a mixture of cupric and cuprous chlorides, the latter, though nearly insoluble in water, being held dissolved by the chloride of sodium. In this process the dissolved iron is separated as hydrous ferric oxide retaining a small portion of ferric chloride. To prevent the loss of chlorine thereby, the use of sulphurous acid to reduce and dissolve this oxychloride of iron was then proposed, but it was found in practice with roasted sulphuretted ores that the ferrous salt formed in the reduction by metallic iron of the cupric sulphate thus obtained renders unnecessary the use of sulphurous acid. In this method with ferrous chloride, the resulting solution being neutral, the absence of arsenic therein is insured. Any silver contained in the ores is chloridized by the action of the copper salts, and made soluble in the bath of chloride of sodium, from which, however, its separation presents considerable difficulties; the precipitation of silver therefrom by metallic copper in the presence of chlorides requiring that the whole of the dissolved copper present should be in the cuprous form. For the rest, the copper is readily precipitated from these solutions by metallic iron.

In the second method referred to above, beginning with a neutral solution of copper sulphate, there is added so much of common salt or other soluble chloride as will serve to convert the copper present into cuprous chloride (58.5 parts of sodium chloride to 63.4 of copper). Through the clear, hot solution is now drawn or driven sulphurous acid gas, got from roasting sulphurous ores, which serves to convert the dissolved copper into insoluble cuprous chloride, with liberation of the previously combined sulphuric acid, and the generation of one-half as much more of the same acid by the oxidation of the absorbed sulphurous gas. The clear acid liquid drawn from the cuprous chloride is then saturated with copper from oxidized ores, and the precipitation by sulphurous gas is repeated indefinitely. The insoluble cuprous chloride obtained in this process may be either reduced

to the metallic state by iron, under water, or decomposed by milk of lime with formation of cuprous oxide and chloride of calcium; the resulting chlorides in either case serving for the chloridization of the further portions of dissolved copper. By the use of a solvent containing only small portions of soluble chloride, any silver present in the ores is chloridized, but remains undissolved in the residue, and may be extracted therefrom by solution, by amalgamation, or by smelting; whereas, in the first process it is held in solution, though, from the presence of the two chlorides of copper, it cannot be readily separated either as iodide of silver or in the metallic state. It should be further said that by the use of an acid solvent for the copper oxide in the second process there is incurred the risk of bringing arsenic into solution should this be present in large quantities in the ore.

To meet the difficulties presented by the previous methods, Mr. Douglas and myself have lately devised and patented a new process. By attacking the oxidized copper ores with a solution of common salt and ferrous chloride, the latter is decomposed, and the resulting clear and neutral solution, separated from ferric oxide, but containing cupric and cuprous chlorides, is next treated with sulphurous acid gas to convert the former into cuprous chloride with separation of free acid. From such an acid cuprous solution any silver present is now readily and completely separated by metallic copper, after which the whole or nearly the whole of the dissolved copper may be precipitated by metallic iron, care being taken to arrest the process before the free acid begins to attack the iron. In this way a solution is obtained containing, besides the regenerated ferrous chloride, a considerable amount of acid. This solution is used to attack a fresh portion of oxidized copper ore, which first neutralizes the acid, and then decomposes the ferrous chloride, the ferric oxide from which carries down any arsenic that may have been previously dissolved by the acid. The neutral solution of the copper salts thus obtained is then again treated with sulphurous acid, and the steps, as above described, are repeated indefinitely.

SILVER INGOT MELTING AT THE MINT OF THE UNITED STATES AT NEW ORLEANS.

BY F. F. CLAUSSEN, ASSAYER OF THE UNITED STATES MINT AT NEW ORLEANS, LA.

(Utah and Montana Meeting, July, 1887.)

THE method of making silver ingots in use at this Mint being radically different from that employed at any other Mint of the United States or, so far as known to me, any Mint in the world, there may be some interest in a description of the apparatus used and a brief history of its distinguishing feature, viz., the rotary ingot-mould, which, though not a recent improvement, is comparatively unknown.

The furnaces used for silver-ingot-melting are four in number and have a capacity of some seventy thousand standard ounces of silver per day of eight hours. This output is increased, when the exigencies of the service demand it, to one hundred thousand ounces. They are built of fire-brick and elevated 2½ feet from the floor of the melting-room, in order to have space to place iron boxes and to work, in case a crucible breaks in the furnace and its contents run into the ash-pit. The furnaces are circular in horizontal section, 2½ feet in diameter and 2¼ feet high inside in the clear, and rest on cast-iron plates, with top-plates of the same, and sides encased in boiler-iron. The furnace-covers are also of cast-iron, dome-shaped, and are handled by means of a small crane and lever. The furnaces stand in sets of two, and one crane and one ingot-machine serves two furnaces.

The crucibles used are made specially for this purpose of a mixture of black-lead and fire-clay; the size known as No. 200 will contain and safely melt, without danger of slopping over, from 8600 to nearly 10,000 ounces of standard silver; and this is the usual weight of the ingot-melt at this Mint, as against 1800 ounces at the Carson City mint and 2600 to 3000 ounces at Philadelphia and San Francisco. Each crucible will endure, on the average, ten melts.

The crucible stands in the center of the furnace, on the bottom of an old one, cut down to a height of about three inches. The fuel used is coke. Experience has shown that, notwithstanding the higher first cost, Connellsville coke is cheaper than Alabama coke and much more suitable to the work. A small triangular black-

es of silver, is used as a pour-

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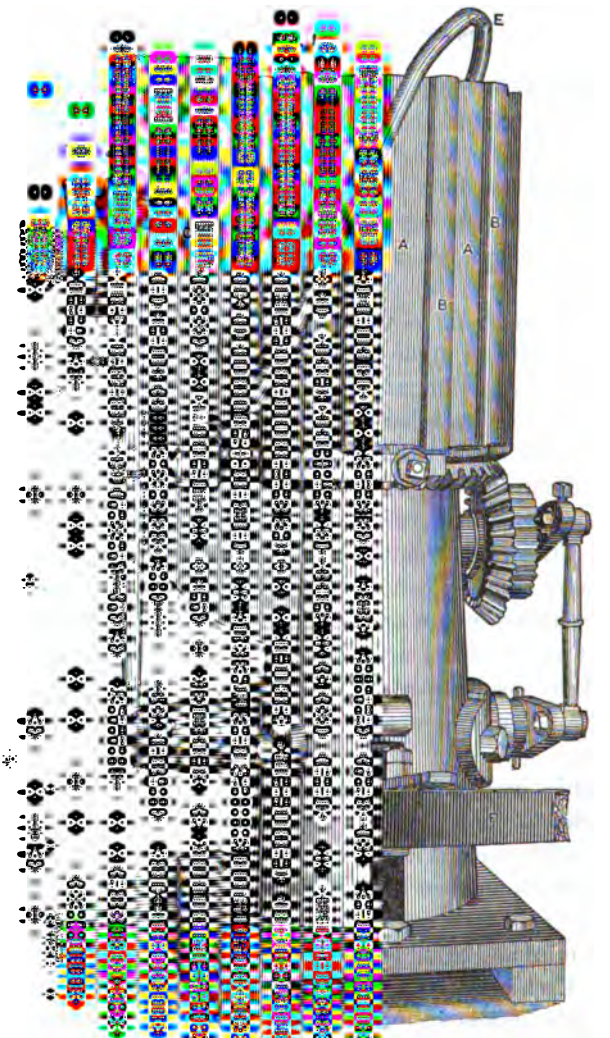

Front View.

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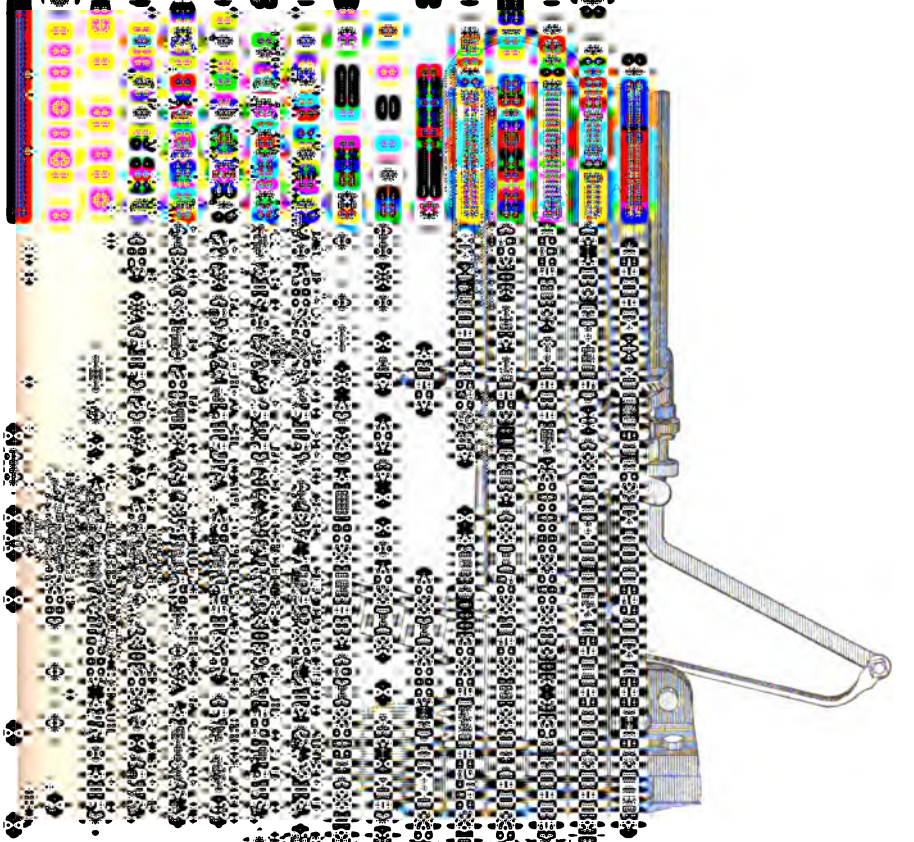

nder. The slots
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cylinder and block are kept cool by a half-inch stream of cold water E, constantly flowing through them. This cooling is rendered necessary by the strong tendency of silver-copper alloys to segregate, and can only be partially overcome by cooling the ingot as quickly as possible after casting. The machine is operated by a foot-treadle (F, shown in Fig. 2, but omitted from Fig. 1 because the photographic original distorted it), and is as follows:

The workman presses down the treadle, which first retracts the block from its position in contact with the lugs on the cylinder, where it has been left during and after the pouring of the preceding ingot; then rotates the cylinder one-twelfth of a revolution, bringing one of the moulds on its circumference exactly opposite the block; then locks the cylinder in place; and finally releases the block from its retracted position, and allows it to be forced (by means of powerful steel springs) firmly against the lugs on the cylinder. So close a joint is obtained that no silver escapes through it, and only a slight feather-edge forms on the ingot after a cylinder has been in use for months. The silver is now poured into the open top of the mould, where, owing to the cooling by water, it immediately sets. The cylinder is again revolved as before, and the ingot at a dull red heat is exposed to view. This ingot remains in the mould until the third or fourth succeeding ingot is poured, when it is removed and placed in a cast-iron box.

This machine was invented by Dr. J. L. Riddell, the Melter and Refiner of this Mint from 1839 to 1849. Dr. Riddell had already achieved distinction by the invention of the binocular microscope; and his desire to improve the slow, laborious and expensive method of casting ingots in hand-moulds, then universally practiced, resulted in this simple and efficient machine. The idea was developed about the year 1844; but from various causes, among others the expense incidental to the proposed change, and the fear expressed by Mint officials that the machine would not prove successful, nothing was done until 1847, when the Coiner, Mr. Philos B. Tyler, resigned to accept the superintendency of the American machine-works at Springfield, Mass. On his departure, he agreed to build a complete ingot-machine, as proposed by Dr. Riddell, for $500. But Mr. Tyler himself possessed a most ingenious mind. He was the inventor of the Tyler cotton compress; had built the first steam coining-presses used in America, and much other Mint and general machinery. Hence it naturally followed that his mechanical ability perfected the crude ideas of his colleague; and the machine, as built

in 1847 and in use here for some years afterwards, was really the joint invention of these two gentlemen. It appears to have been successful from the first. The original machine was in use here for many years; the hand-moulds were immediately discarded and have never since been used in this Mint for casting silver-ingots. Dr. M. F. Bonzano, the able and efficient officer who succeeded Dr. Riddell as Melter and Refiner, made several important improvements to the machine during his term of office, which extended from 1849 to 1883.

The advantages derived from the use of the rotary ingot-machine are obvious. Only four large furnaces are required, instead of a large number of small ones; the number of workmen required is also greatly reduced; and the weight of each melt is increased to an average of 8600 standard ounces, while by the hand-mould method the limit appears to have been reached at 3000 ounces, and the average weight by that method was about 2600 ounces.

An objection is urged against making so large a melt of standard silver, based upon the supposition that the alloy will vary in fineness between the first and last ingots or from top to bottom of the melt, the difference being caused either by segregation or by an increase in fineness caused by a portion of the copper being burnt out, during the relatively long exposure of the melt in the furnace. Both these sources of error are avoided in our practice: the first, by a thorough mixing just previous to the pouring; and the last, by keeping the melt constantly covered with a thick layer of finely ground charcoal, and by the use of the ingot-machine, which enables us to take out a melt in twenty minutes. Years of successful practice have shown that no greater deviations of fineness exist in these large melts than in the smaller ones.

The Melter and Refiner bases his calculations for the amount of copper and silver necessary to form ingots conformable to law, upon the report of the Assayer of the fineness of the bullion, and makes up his melt accordingly. As the ingots made from fine silver and copper alone are not quite as ductile as those containing a certain proportion of previously melted alloy, it is customary to add to each melt a certain amount of "clippings" and "planchets." Clippings are the remainder of the ingots after the blank coins have been cut from them, and planchets are the blanks. A melt, as sent to the melting-room, will be composed about as follows: 5835 ounces of fine silver; 3000 ounces of clippings and planchets; and 577 ounces of copper. All calculations for melts are made independently

and in duplicate by two clerks, in order to avoid error; the bullion and copper are exactly weighed in ounces and decimals for each melt, put in boxes and taken to the melting-room, where the furnace and crucible are already hot. The bars of fine silver, weighing about 1200 ounces each, are carefully lowered into the crucible with a block and tackle. Two bars on end will fill the bottom of a No. 200 crucible. The interstices are then filled with clippings, another bar is put in and clippings and planchets are added until the pot is full. When this is melted down, the silver is "thickened" by adding clippings until they appear above the molten silver; this forms a bed to receive the remaining bars, and prevents splashing, should the bars slip from the tongs in filling. The remaining bars and clippings are then added. When all the silver is in and thoroughly melted, the copper is added. The furnace is closed again for a short time, and the fire urged, in order to get a hot and fluid melt. When this has been secured, the furnace and crucible are again opened, the previously-heated stirrer is introduced, and the whole mass is thoroughly mixed by churning the stirrer up and down some four hundred times through the molten alloy. Count is kept, and all melts are stirred in this manner. From the time that the silver was first melted it has been constantly covered by a coating of charcoal-dust, renewed from time to time as it is burned off. During the stirring the amount used for covering is increased; but after this operation is finished, the melt is skimmed so that only a thin layer remains. Care is taken, however, that the surface is kept always covered, up to the pouring of the last ingot. The stirring being completed, the pouring begins by the melter grasping the hot pouring-crucible in his long-handled tongs and ladling out the silver. The first dip of about six ounces is poured into a copper cup containing ice-water, which granulates the alloy. This sample is numbered and set aside for assay. The next dip is cast as the first ingot. During the pouring five samples are taken for assay: one at the beginning, one at the end, and the other three at intermediate stages, so as to insure a fair sample of the whole melt. These are assayed separately; and when there is a greater difference between the first and last granulations than three-tenths of a millième, the whole melt is condemned and remelted. Such is the thoroughness of the work that but very few melts are condemned for this cause. Although the law allows a variation, in the case of silver ingots, of three millièmes from the standard of 900, it is the practice at this Mint to condemn ingots when below 898½ or above

900½, the aim being to hold the coin as issued as close to the lawful standard as possible.

The pouring being completed, the ingots are spread upon the floor to cool, after which they are "topped" and the edges are filed, so as to leave a smooth, clean ingot with a solid top. If passed by the Assayer they are delivered to the Superintendent, to be formed into standard silver dollars.

SILICA DETERMINATIONS IN BLAST-FURNACE CINDER.

BY CLEMENS JONES, B. S., HOKENDAUQUA, PA.

(Duluth Meeting, July, 1887.)

AN interesting paper by J. E. Merion and Edward Hart, in Volume I., No. 2, of the *Journal of Analytical Chemistry*, on the Decomposition of Blast-Furnace Cinder by Acid, describes a plan of sampling the cinder by means of a previously-heated iron bar thrust into the cinder car or into the cinder at the notch, and then allowed to cool, thus chilling the cinder which adheres to the bar, and rendering it decomposable by hydrochloric acid, of 1.12 sp. gr. A short series of results is given, from which the deduction follows that blast-furnace cinder "*is* decomposed perfectly when chilled."

Where many silica determinations are to be made, this method, if reliable, would effect notable economy of time and reagents. The conversion of insoluble silicates by fusion with an alkaline carbonate, together with the subsequent solution, evaporation, drying, etc., is destined to remain the standard method. But all fusion-methods are time-consuming and expensive. Chilling appears to render the supplementary method available for all practical purposes, in the determination of silica in cinder.

But, with a view of securing more evidence on the subject, and in view of the fact that uniform results were not obtained by employing this method, it was decided to test the method in the laboratory of the Thomas Iron Company where a large number of silica determinations are made. The experience met with may not be devoid of value, both theoretical and practical.

Mr. Iles's paper, in the *Chemical News*, Volume L., 1884, doubtless suggested to several chemists, connected with iron metallurgical

works contemporaneously, the applicability of chilling the cinder, for inquiry elicits the information that, in two instances, it was known at least two years ago.

About the same time, this subject was carefully considered, with reference to the adoption of a quick and safe way to sample cinder, at a number of furnaces located in different places. Although the "bar method" was then known to be in use, it was rejected in favor of the plan then devised and used now, which is simple, reliable, expeditious, and produces exactly the same effect on the cinder.

A cast-iron pan, similar to an assay-pan, 5 inches square, and having in it four shallow, hemispherical cups, is riveted to a handle 20 inches long. The cups have the furnace-numbers countersunk in the bottoms. The sample is taken by filling this pan directly, or by means of a ladle, with the fluid-cinder as it flows from the cinder-notch. It chills instantly, and by the time it can be carried to the laboratory, is ready for treatment. By this system, there is no danger of confusion, and there are four small cakes from which to choose a final sample.

In the experiments made, the cinder was sampled at all places in this way, and the method, with given modifications, was applied to cinders from twelve blast-furnaces, thereby securing results by identical procedures upon cinders of varied composition, which offered, by reason of different ores and dissimilar conditions, all of which are known, a liberal basis for conclusions.

It may be noteworthy, that in January, 1887, the experiment was made of causing the cinder from the runner to fall into water, with the expectation of simplifying the sampling, and obtaining a decomposable product. The result was not unfavorable, but showed no improvement on the practice in vogue. The two results here given are the only records found of the experiment:

Furnace.	By fusion with Na_2CO_3. SiO_2 per cent.	By decomposition with HCl of 1.12 sp. gr. SiO_2 per cent.
No. 2,	44.13	41.53
No. 1,	41.03	40.63

In the presence of alkalies in appreciable amount, this would be unreliable, and it would not answer at all for basic cinders. A fine jet of steam would probably make a better sample.

The method of fusing used is as follows: weigh 0.5 grm. finely powdered cinder, transfer to platinum crucible (30 c.c.), mix with $3\frac{1}{2}$ times its weight of Na_2CO_3; replace cover, fuse over blast-lamp, cool by turning off flame and turning on blast. Place in porcelain

dish (300 c.c.) in which there is 100 c.c. water, cover with inverted funnel of smaller diameter than dish, add 40 c.c. HCl of 1.12 sp.gr., wash out crucible, evaporate to dryness in steam-bath, dry 1½ hours at 1.20° C., take up with 50 c.c. HCl, wash, add water, filter, and ignite.

The following are the results of comparative tests made April 19th, 1887.

Furnace.	Fusion method. SiO_2 per cent.	HCl method. SiO_2 per cent.
A,	40.99	40.92
B,	42.83	43.33
C,	44.37	43.98
D,	42.17	41.58
E,	44.53	44.34
F,	41.89	42.59*

A variation of about 0.5 per cent.

The HCl method was practiced as follows: weighed 0.5 grm., transferred to porcelain dish, stirred with water (50 c.c.), added 25 c.c. HCl of 1.12 sp.gr., evaporated and then treated as above.

Determiuations were repeated with cinders from the same furnaces, with the following results, May 2d:

Furnace.	Fusion method. SiO_2 per cent.	HCl method. SiO_2 per cent.
A,	42.27	42 99
B,	39.07	39.87
C,	42.77	42.63
D,	43.57	44.71
E,	43.41	44.37
F,	40.67	40.87

Not very fairly agreeing results. We observed on the porcelain dishes in which the HCl method was used a coating only removable by scraping with a platinum knife.

We next treated the cinder with acid in platinum dishes, May 9th.

Furnace.	Fusion method. SiO_2 per cent.	HCl method in Platinum. SiO_2 per cent.
A,	43.32	44.50
B,	40.88	42.26

A marked disagreement, but showing that by the fusion method the porcelain is not attacked.

Since fusing cinder in platinum is eventually severe on the crucibles, we next tried, May 9th, to fuse in nickel crucibles.

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* There was some Fe_2O_3 in this.

Furnace.	Fusion in Pt Crucibles. SiO_2 per cent.	Fusion in Ni Crucibles. SiO_2 per cent.
G,	40.18	40.17
H,	38.84	39.76
I,	40.74	41.10
J,	43.44	43.42
K,	44.68	45.84
L,	44.43	43.90

This was found to be unreliable, the discrepancy increasing with repeated trials in nickel crucibles.

Platinum dishes were again tried, May 16th, in the HCl method; this time evaporating rapidly.

Furnace.	Fusion method. SiO_2 per cent.	HCl method in Platinum. SiO_2 per cent.
A,	42.10	42.58
B,	43.14	42.50

With the exercise of all possible care, there is still unavoidable variation.

Selecting titanic acid as the probable *bête noir*, the experiment was made, May 16th, with cinders known to contain it.

Furnace.	Fusion method. SiO_2 per cent.	HCl method in Porcelain. SiO_2 per cent.	
G,	40.80	40.10	Took up with 50 c.c. boiling HCl of 1.12 sp. gr. and heated to boiling 3 minutes.
H,	40.08	40.25	
J,	41.28	43	
L,	41.04	42.75	

Our surmise seemed justified by the differences. The 42.75 silica from L furnace, which was known to carry the most TiO_2, was then fused with $K_2S_2O_7$, dissolved with cold water, filtered and again weighed with this result: silica, 40.588. Titanic acid was then precipitated in the filtrate. This fact is largely at variance with Messrs. Merion and Hart's experience, and seems to indicate that titanic acid is a source of dangerous error.

Boiling HCl as used in the above test was again tried, May 23d, in the HCl method, with these results:

Furnace.	Fusion method. SiO_2 per cent.	HCl method. SiO_2 per cent.
A,	46.84	47.42
B,	41.78	45.96
C,	41.28	44.10
E,	39.72	38.72
F,	35.22	35.82
G,	39.92	40.28
H,	35.82	37.60
I,	42.54	42.74
J,	40.38	42.02
K,	45.02	44.68
L,	40.62	41.24

Giving large disagreements. These suggested a comparison, June 9th, of the two methods, with fusion of the silicas giving high results by the HCl method.

Furnace.	Fusion method. SiO_2 per cent.	HCl method. SiO_2 per cent.	Fusion of the latter. SiO_2 per cent.
A,	41.62	42.34	39.24
B,	35.62	37.68	36.38
C,	43.72	44.14	41.32
E,	42.96	44.16	42.66
F,	42.06	40.20	
G,	38.64	38.06	
H,	38.04	38.98	36.84 (?)
I,	42.64	40.54	
J,	39.66	38.82	
K,	42.16	43.46	41.94
L,	36.06	41.06	36.32

These results were not satisfactory from any point of view, but are here inserted for the sake of the variations and the lessons they may teach. The work was done by assistants in the laboratory.

A lot of basic cinders were next tried with the comparison as follows:

	Fusion method. SiO_2 per cent.	Acid method. SiO_2 per cent.	Former determination. SiO_2 per cent.
No. 1,	33.08	34.28	32.56
No. 2,	36 22	34.72	32.54
No. 3,	32.44	33.34	32.04
No. 4,	33.06	33.12	32.24
No. 5,	34.48	33.44	32.66
No. 6,	35.42	34.32	29.88
No. 7,	34.24	34.00	29.12
No. 8,	32.56	31.92	31.52

The agreement between the first two columns is here pretty good. The column of "former determinations" gives the results as obtained at the works from which the samples came, of which the head-chemist writes: "Ours were only furnace-checks done by a boy and very rapidly." . . . . "The samples were all chilled." This conforms with our experience above cited.

I think a just summary of these comparisons is that the acid method, when employed upon chilled cinders and with proper precautions, is safe for all practical purposes. But chief among the precautions is a re-determination by the standard method in cases of marked variation in consecutive tests. Clearly the acid method is prone to irregularities, caused by titanic acid and variable cinder composition. Moreover, it is probable that by drying at 120° C., part of the alumina is rendered insoluble.

A trial was made, July 25th, of the acid method as here described, for comparison with the method employed in Messrs. Merion and Hart's experiments. The results in duplicate show the close agreements obtained by drying at 120° C.

	SiO_2 per cent.
E. Furnace fusion method,	44.22
By HCl drying at 120° C. for one hour, No. 1,	44.04
" " " No. 2,	44.04
By HCl with evaporation alone, No. 1,	43.74
" " " No. 2,	43.52

That the results by this latter method are really too low, is shown by the following comparison made August 9th:

Furnace.	HCl method. SiO_2 per cent.	HCl method with drying. SiO_2 per cent.	Fusion. SiO_2 per cent.
A,	40.70	42.42	41.01
B,	39.34	41.26	38.90
C,	41.42	43.04	41.80
D,	39.58	39.44	41.04
E,	41.30	42.12	
F,	41.44	40.82	
G,	34.86	35.42	
H,	38.16	39.04	
I,	37.78	38.50	
J,	36.52	36.68	
K,	38.38	40.86	

In support of this testimony, the results obtained September 4th, by employing the acid method without drying, and then evaporating the filtrate and treating as usual for silica, are appended:

Furnace.	HCl method. SiO_2 per cent.	Found in filtrate. SiO_2 per cent.
A,	38.90	0.10
B,	39.10	0.09
C.	40.10	0.12
D,	37.30	0.07
E,	37.78	0.11
F,	40.	0.09
G,	35.60	0.09
J.	38.92	0.22
K.	37.30	0.20
L,	39.68	0.17

The first four of this series, tried by the fusion method and the acid method, with drying, gave the following:

Furnace.	Fusion. SiO_2 per cent.	Acid with drying. SiO_2 per cent.
A,	38.66	38.70
B,	39.88	39.14
C,	41.04	41.44
D.	37.02	38.46

Some few discrepancies, not here accounted for, might raise the question, whether the samples were all chilled; but that is settled by their physical properties. They were all brittle, vitreous, and, in most cases, translucent. The system of sampling is applicable to both acid and basic cinders.

It may be urged, that no value attaches to technical results in investigating a method; but, assuredly, a technical method requires its limits of variation to be established in the way it is to be practiced.

The relation between a variable composition of slags and their decomposition with hydrochloric acid, is a subject still open for examination. The molecular action of chilling might not be inconsistent with the analogy of the natural silicates.

The acid method, with drying for one hour at 120° C., certainly gives better results than without drying, and is unaccompanied by the error due to loss by drying quickly at a high temperature after evaporation, which often, in the hands of an assistant, may be incompletely done. Laboratory attendants are parts of the technical apparatus, and are too often left out of the calculation.

But all methods so much hastened as to cause mechanical loss ought to be discouraged and abandoned. It may be said, that approximate results only are sought, but, such approximation is but guessing under another name; and it may certainly be anticipated that a method which, even under special care, unavoidably involves a perceptible loss, will be entirely unreliable in the course of routine analysis. The result will be left to fate, which will weigh the precipitate in the balance and find it wanting.

THE KAIPING COAL-MINE, NORTH CHINA.

A REPORT OF KWONG YUNG KWANG, ENGINEER AT THE MINE.

REVISED AND PRESENTED BY J. M. SILLIMAN, LAFAYETTE COLLEGE, EASTON, PA.

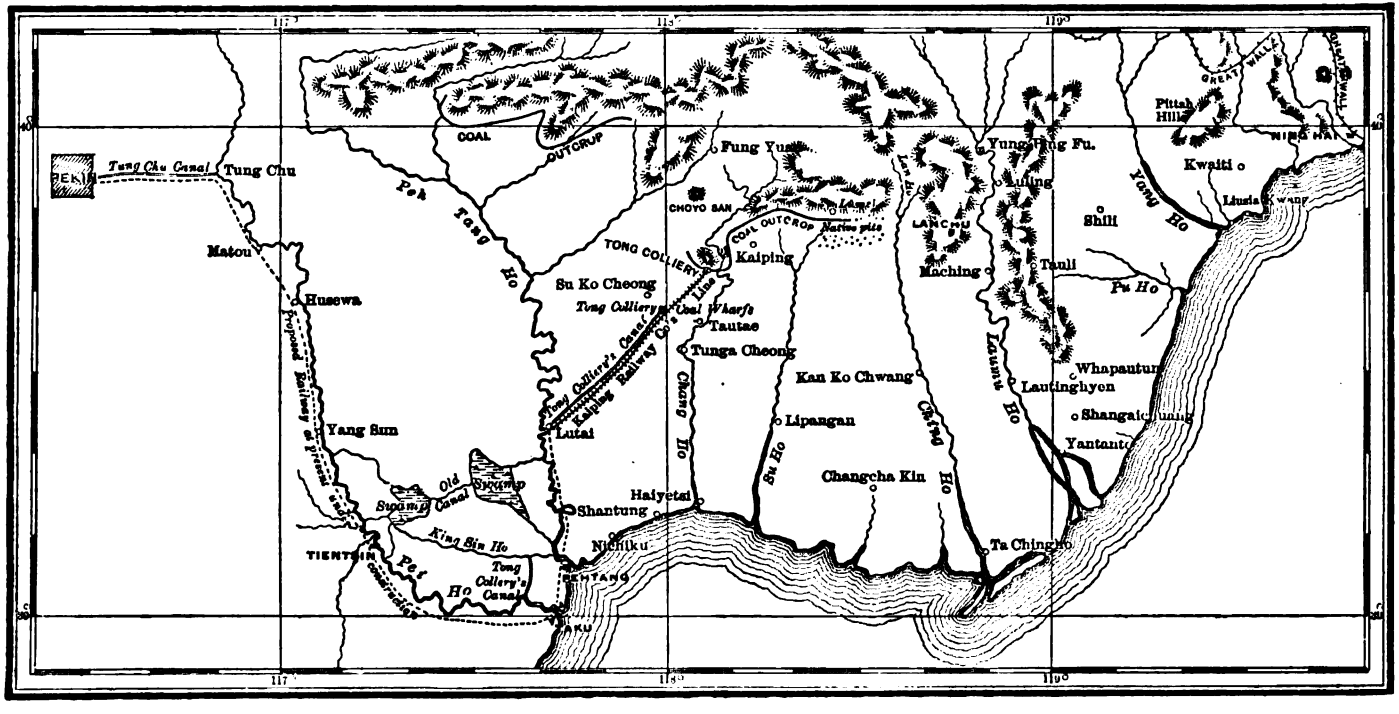
(Duluth Meeting, July, 1887.)

THIS mine, which is sometimes called Tong colliery, from its director, Tong King Sing, is about 80 miles northeast of Tientsin (see map, Fig. 1).

Fig. 2 shows a vertical transverse section through the coal-beds. These beds have a dip to the south of about 45° and are interstratified with sandstone, shale, indurated clay and fire-clay. Of the

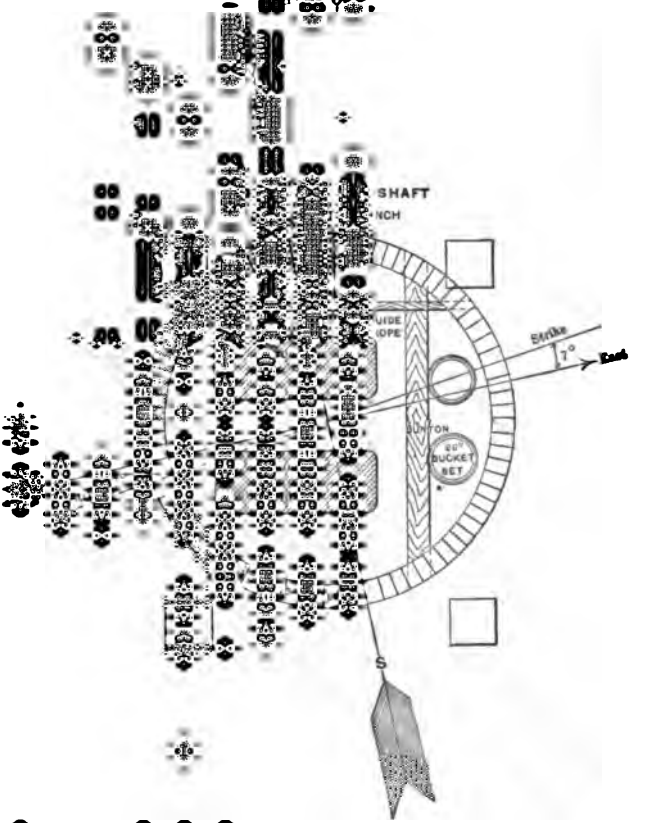
FIG. 1.

MAP OF KAIPING COAL FIELD, NORTH CHINA.



No. 12 Seam
No. 9, 10 Seams
No. 4 Level
Dip Workings
No. 5 Level
A
Sump

[redacted] workable. The coal is bitumi- [redacted] d by natives west of Pekin.) [redacted] y fire-clay, which is full of



[redacted] it to have been the ancient

[redacted] E COAL.

[redacted]	Moisture. per cent.	Sulphur. per cent.	Iron. per cent.	Volatile Mat. per cent.
[redacted]	.56	.29	1.47	29.60
[redacted]	.30	.75	...	27.45
[redacted]	.65	.97	...	26.33
[redacted]	.40	1.86	...	25.16
[redacted]	.47	.986	1.61	29.53

The Shafts.

In the year 1879, after a few trial borings with the diamond drill, two shafts, each 14 feet in diameter, were sunk at a distance of 100 feet from each other. (See Figs. 3 and 4).

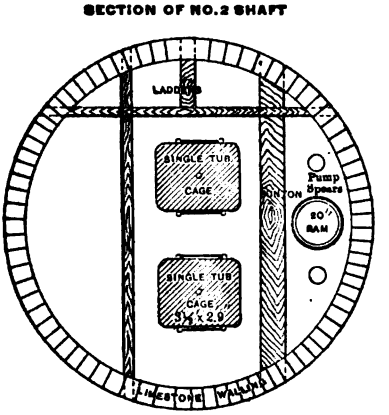
Shaft No. 1, the downcast, was sunk to the depth of 560 feet, and No. 2, the upcast, to the depth of 300 feet.

They are walled with limestone of the following dimensions: For the first 24 feet the sections are 24″ x 9″ x 9″, the next 25 feet 15″ x 7″ x 7″, and from this point to the shaft bottom 12″ x 5″ x 5″. The stones are all cut to fit the circular form of the shaft, a block of the first section costing but 5 cents. The compartments of the shafts for winding and pumping are shown in Figs. 3 and 4.

Methods of Working.

As soon as the shaft reached its required depth a cross-cut was driven north to cut all the coal-seams, and when any seam, as, for instance, A, was reached (Fig. 2 and No. 5 seam, Fig. 5*), it was

Fig. 4.



followed along the strike east and west. After extending these levels on each side of the cross-cut a distance of 50 feet, winzes R′,R′ (Fig. 6) were driven up to No. 1 level for ventilation. These winzes at a distance of 20 feet above No. 2 level are connected by a ventilation cross-head. As the drifts advance, winzes are thus

* In studying Fig. 5, it should be borne in mind that the heavy black lines indicate the drifts or gangways, not necessarily, the coal. Thus, it will be seen that by reason of faults, throwing the seams, the gangway which is at the main stone-drift, in No. 5 seam, is found at one end in No. 7, and at the other in No. 3.

No. 7 Seam
No. 5 Seam
GREAT
No. 2 Shaft
No. 2 Sh. ft
STONE DRIFT
No. 3 Seam
No. 5 Seam
Surface Packing Incline
Main Coal Incline to No. 3 Level
GREAT
No. 3 Level
No. 3 Seam

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R′,R′ are stone-

e Fig. 7). Coal-
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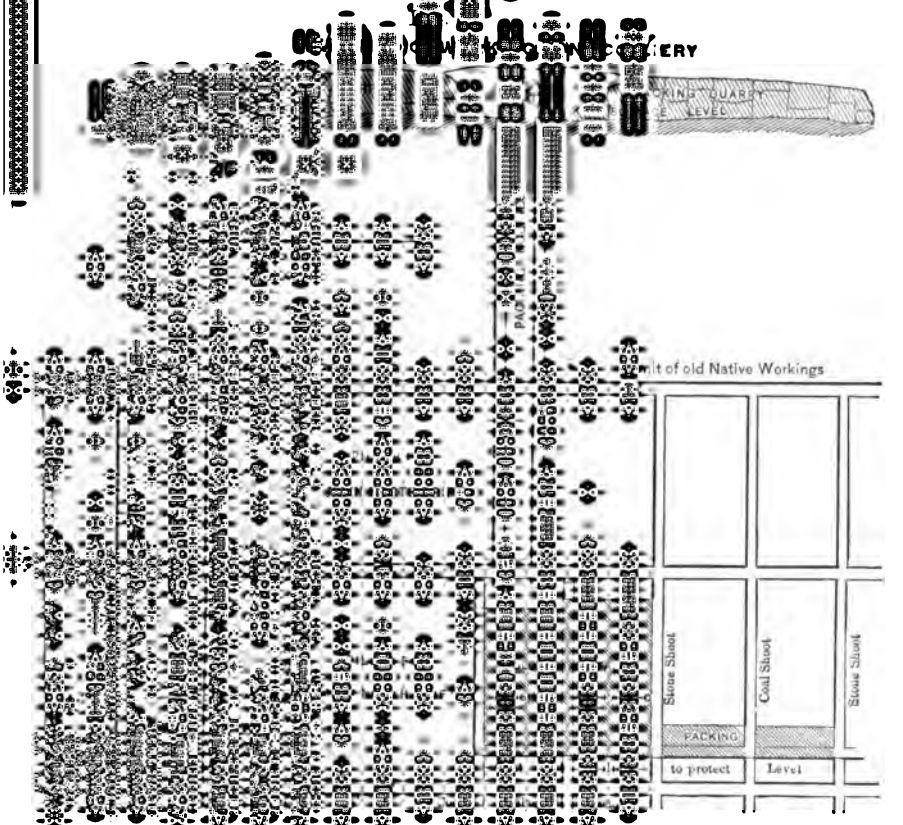
with No. 1 level.



rom the ladder-
miners begin at
Round props of
coal as the cutting
y the coal is filled
m the surface to
in packing-tubs
inclines packing-

ls and emptied

v baskets which
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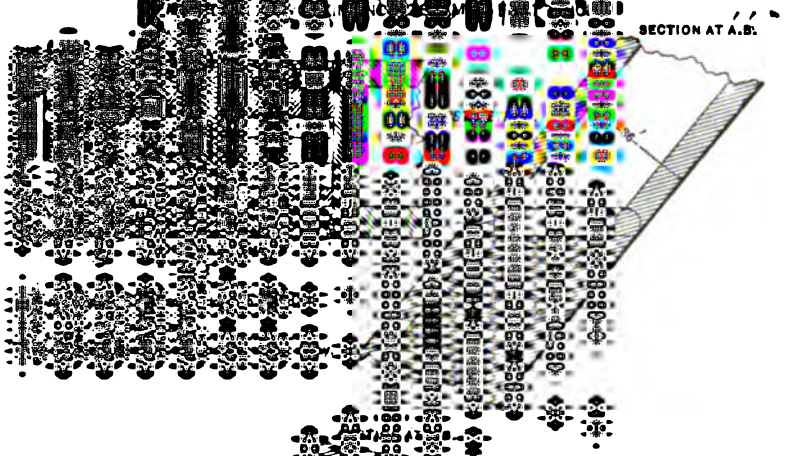
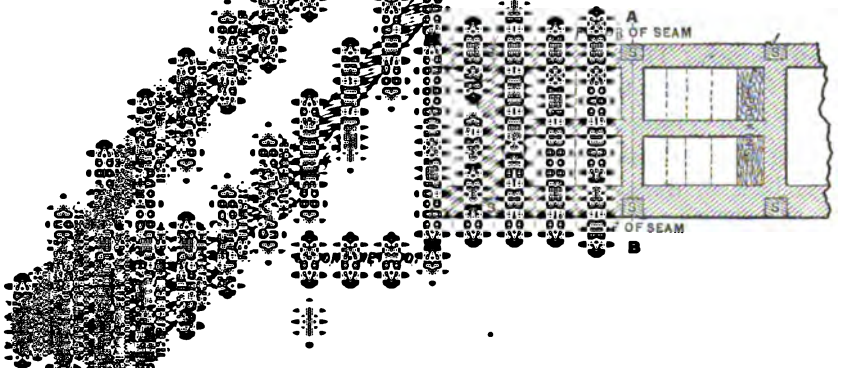
slice is thus re-
cted by build-
3-inch plank as





gh these shutes
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the working of
(Fig. 8). This is

divided into five slices parallel to the stratification, which are worked off successively in ascending order like so many district seams of 7 feet thickness. The average cost of packing per ton of coal is 4 cents. In the thin coal-seams this packing method is so costly that the long-wall method is adopted. With this packing system very little timber is required, and the surface is preserved intact.

Haulage.

This is accomplished solely by mules and ponies. Stables are provided for these in each of the three levels. At present there are 40 mules and ponies working during the three shifts, each animal pulling about 8 coal tubs. A good mule and pony costs about $30 and $20 respectively. When the workings become extended a mile or more from the shaft-bottom, a main- and tail-rope system of haulage will be adopted.

Winding.

All the coal from No. 1 shaft-bottom is brought to the surface by a pair of horizontal, non-condensing steam-engines of 22-inch cylinder and 4½ feet stroke. The winding-rope is 1¼ inch steel, weighing 7 pounds per yard, and with a working strength of 6 tons. The winding-drum is 18 feet in diameter, and is lagged with hard wood. The double-decked cages of No. 1 shaft are suspended from the winding-rope by six ¾-inch chains. Each cage carries four coal-tubs which, when full, weigh about two tons each, while the empty cage weighs about 22 cwt. The cages run on steel guide-ropes of 1-inch diameter. The head-gear pulleys have a diameter of 12 feet.

No. 2 shaft is provided with a pair of small winding-engines of 10-inch cylinders and 20-inch stroke, the engine connected by gearing with a drum 8 feet in diameter. The steel winding-rope of 1 inch in diameter has a working strength of 3½ tons, and weighs about 4 pounds per yard. The cages, each weighing about 8 cwt., and carrying one tub, are suspended by four ½-inch chains.

Pumping.

Fig. 2 shows the system of mine-drainage. The water is taken to the surface in two lifts. First, the water in No. 5 seam dip-workings is hauled up to No. 3 level in water-cages by a pair of winding-engines of 18-inch cylinder and 30-inch stroke. From this point (B) the water is lifted 260 feet to a wrought-iron tank (C) communicating with a stone-drift (D), which runs into the sump

urface by a ram.
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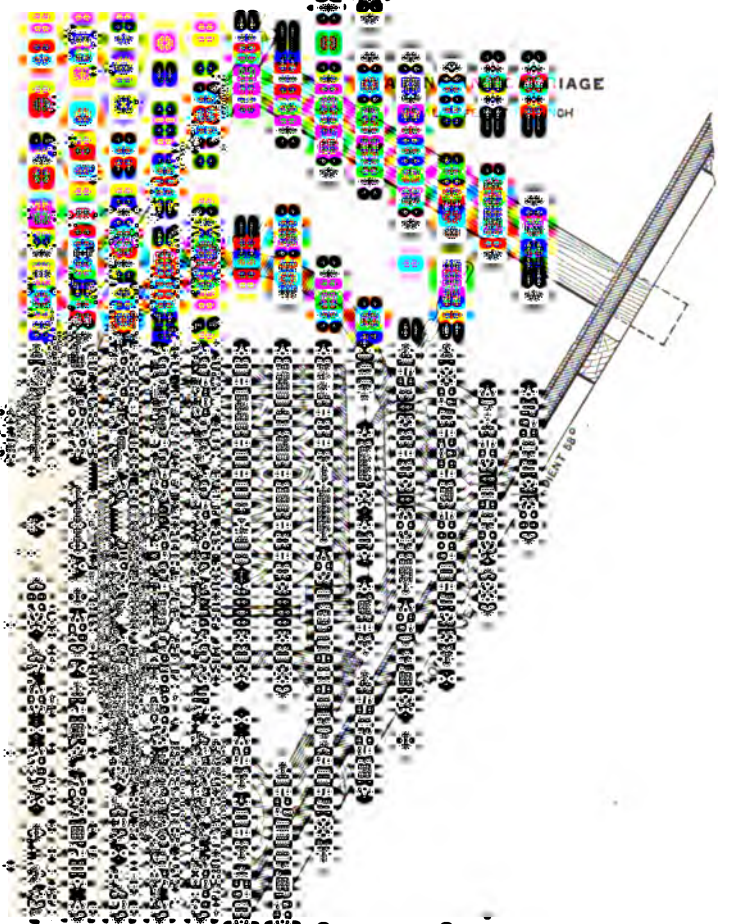
w-pressure cylin-



of 6 feet per min-
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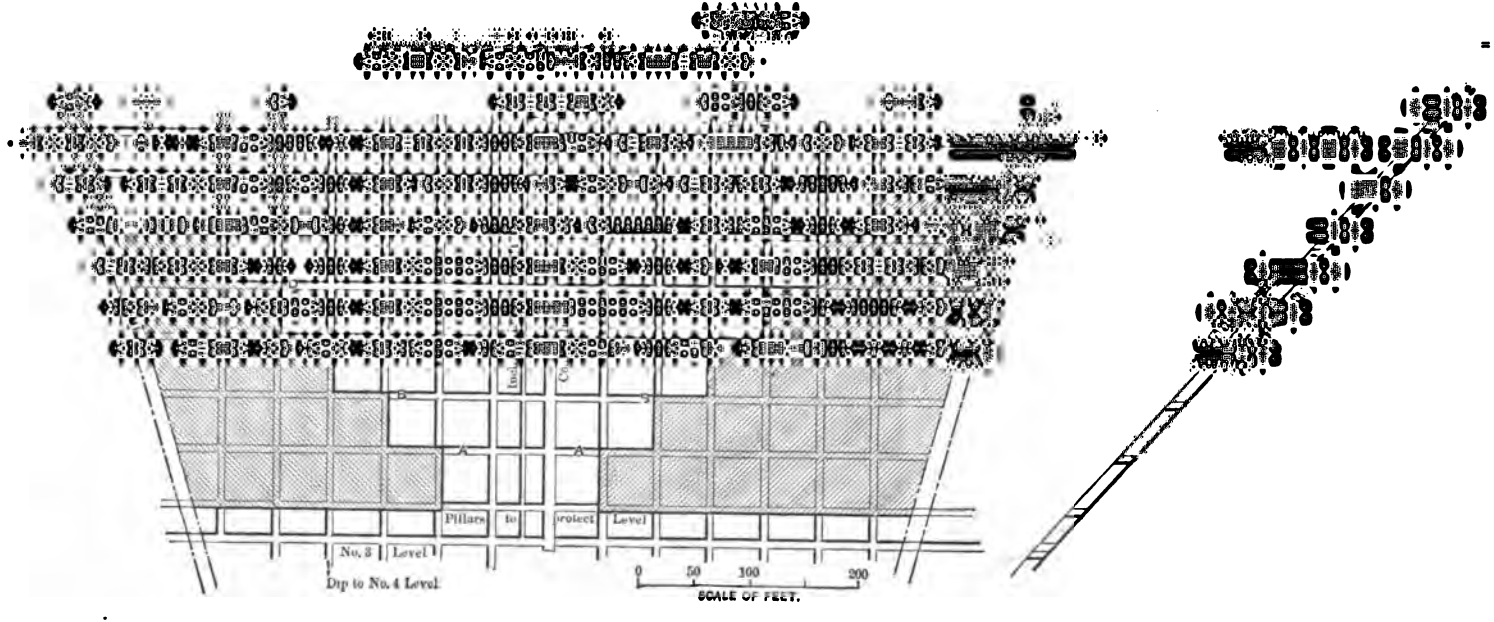
square feet. The


B
C
A
A
Pillars
to
project
Level
No. 3
Level
Dip to No. 4 Level.
0
50
100
200
SCALE OF FEET.

fan is now working at a speed of 30 revolutions per minute, with a water-gauge of 0.6 inch, and producing a ventilating current of 120,000 cubic feet per minute. The average velocity of the air through the workings is 5 feet per second. The distribution of the air through the mine is shown in Fig. 2. Candles and open lamps are used in all seams except No. 5, where safety lamps only are allowed. About three years ago a dust explosion occurred in this seam, killing three men and injuring several others.

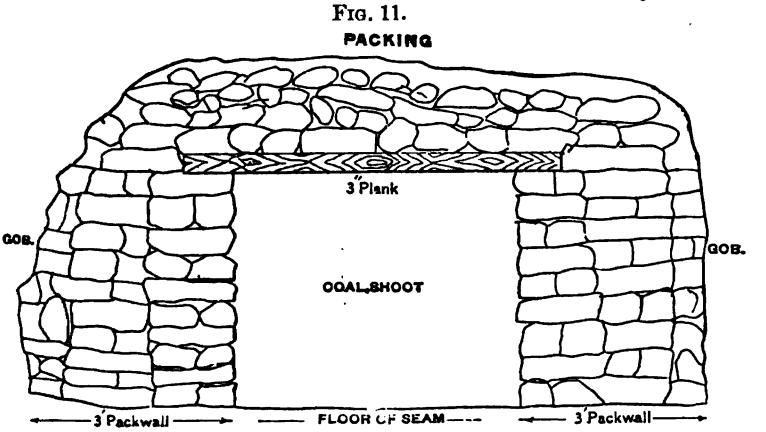
Accidents.

These are occasioned by falls of roof- and coal-pillars; the falling of men down inclines, shafts and stone-shutes; and blasting and dust-explosions. Last year about 13 men were killed in the mine.

Underground Workmen.

The miners belong to the lowest class of society; but, although ignorant, are very obedient and of a kind disposition. They possess many remarkable traits of character. Stealing is considered by them

Fig. 11.



a virtue; and, although they are searched by a special staff of policemen when they come out of the pit, yet they frequently manage to take away some things, such as bolts, nuts, fish-plates, nails, and even brattice-cloth, which they tear away from air-stoppages. In case they are caught, they are not only fined for the cost of the article, but receive a punishment of 100 blows on their backs.

The Chinese miners have not yet appreciated the motto of their foreign brothers that "cleanliness is next to godliness"; for they

wash themselves only once a year. This annual ablution is for the New Year occasion, when they dress up in their fine clothing to receive the New Year greetings of their friends. It would be difficult for a stranger at first sight to determine whether the miners belonged to the Mongolian or the African race.

The men are very fond of sleeping when they ought to be at work. If one is caught in this condition and after being awakened asked why he is not working instead of sleeping, he will face the overseer with a grin and calmly reply that he was not sleeping at all, but was tired and merely sat down to rest himself. The Chinese miners are somewhat superstitious, they believe in evil spirits and bad omens. Last year two accidents happened through this superstitious dread. While a miner was cutting coal up a rise about 100 feet from No. 1 level, he suddenly thought he saw a black wolf, with red eyes, growling at him. In his terror he fell down the winze and was severely injured.

Wages.

Daily wages are paid as follows:

Interpreters to foreign overmen,	20 cents.
Pony and mule drivers,	20 "
Sinkers,	20 "
Masons,	20 "
Engine-men,	20 "
Coal miners,	15 "
Door-boys,	13 "
Switch-boys,	13 "
Drain-boys,	13 "
Drum-boys,	13 "
Cantonese boatswains or pump-men,	$1.00
Chinese pit-deputies,	$1.00

These wages seem very low, but it must be considered that the men do only one-fourth the work accomplished by English miners.

Out-put of Coal.

The daily out-put is 950 tons, which is hauled up in two shifts of 8 hours each.

Market Price, Per Ton.

Best No. 5 Lump,	6 taels = $8.40
Best No. 5 Nut,	4 taels = $5.60
Best No. 5 Dust,	3 taels = $4.20

NOTES ON THE REGION NORTH OF THE VERMILION LAKE DISTRICT, IN BRITISH AMERICA.

BY DR. THEODORE B. COMSTOCK, CHAMPAIGN, ILLINOIS.

(Duluth Meeting, July, 1887.)

In the summer of 1877, the writer led a party of four young men from Cleveland, Ohio, by way of the Great Lakes, to Prince Arthur's Landing (now Port Arthur), at the northwest corner of Lake Superior, thence, *via* the Dawson route and side journeys, as far as Cross Lake. It was the original intention to pass down the Pigeon river to its mouth, but the loss of one member of the party for six days in the forest, added to other obstacles, made it necessary to turn back. A well-filled note-book, together with illustrative collections, were sunk beyond recovery at the head of the Maligne rapids by the breaking of a tow-line, and two of our number (including myself) narrowly escaped drowning. I had, probably, obtained information of considerable economic value, but the necessity of travelling nearly 125 miles for provisions (excepting a meagre supply kindly furnished us by a Canadian government gang whom we fortunately met), rendered further note-taking impossible.

Until now, such results of this expedition as could be recorded from memory have gone unpublished, because of my inability to make them serve any useful purpose. The trip of the Institute to Vermilion Lake, only a few miles south and east of our turning point, gives some little excuse for presenting the following lines at this time.

The ridge back of Port Arthur, Ontario, which follows approximately the course of the Lake Superior shore line, brings up the slates and accompanying strata through which the silver and gold veins of that region have penetrated. The interior district, back from the lake, may be briefly described as a glaciated area, having its general drainage less modified by erosion than by the original structure. This statement, however, must be somewhat qualified with reference to minor details, which all bear the unmistakable impress of the ice-flow and its recession.

In many respects there is much in this tract to remind one of the Sault Ste. Marie topography, excepting that the granitic character of the rocks is here largely replaced by fissile strata.

The Kaministiquia river cuts through several of the parallel folds,

pursuing between them a course which is often nearly the same as that of the ancient glacier. The "Height of Land" is an all but inappreciable water-divide, where the old Dawson route crosses it on the portage between Baril lake and Lac des Milles Lacs. This route, laid out by Wollesley, years ago, is made up of lakes and short portages, following now a somewhat artificial "meander" from stream to stream by way of dammed-up lakes which overflow wide areas. The shore-lines of some of these lakes, as notably that of the lower portion of Lake Kaogossikok, are extremely tortuous, due largely to this overflow against the numerous irregular, low hills, rising like *roches montonnées* along the borders.

It is not until one passes many miles from the Height of Land, going northwestward, that the drainage exposes a section younger than the metamorphic slates. At the Maligne rapids, the river cuts down rapidly through this series, and again runs a few miles sluggishly (owing to another dam) until it plunges suddenly over a thick, tough dolomite formation into an alluvial basin, through which it twists for a long distance. This character is maintained over the Cross Lake region, and only a gradual transition is apparent as one turns southward and eastward to the headwaters of the Pigeon river. Probably some important changes occur along the upper course of Vermilion river, but I was unable to detect from surface exposures any evidence of the existence of large bodies of iron-ore in this particular district which I am describing. Farther back in our course, along the shores of Lake Windigustigon, and, *possibly*, east of the Height of Land, on Lake Shabandawan, there were more favorable indications, but I am now unable to write clearly upon this subject.

The prospects of gold and silver mining are unfavorable, especially in places along Lac des Milles Lacs and east of the Height of Land. In 1877 the Canadian Pacific R.R. was completed only 25 miles from Fort William (32 miles from Port Arthur), but the grade had gone as far as Savanne, which point I also visited. There is a region extending from this point southwestward, off towards Vermilion lake, which will well repay careful investigation. At the time of our visit, its economical development was hindered by the cost of official surveys, owing to its distance from any located Government landmarks. Now this difficulty has been largely overcome, and communication with the railroads north and south is not a very troublesome matter. Good iron-ores occur along the Canadian Pacific; and it is probable that investigation will reveal

valuable depoeits of other minerals in the little-known region west of Cross lake and north of Pigeon river. Much of the territory can be made available for agricultural purposes also, though the climate is one of extremes of temperature.

The timber in some parts was comparatively thick and of excellent quality, in 1877. What is the condition of the forests now, I do not know. Some large tracts had then been burned off through carelessness, particularly one belt fifteen miles long and three miles wide along the northeast shore of Lake Windigustigon.

The only inhabitants over the whole area, beyond the first twenty-five miles from Port Arthur, were a few agents of the Government, an Indian acting as Hudson Bay Co.'s agent at Shabandawan, and three or four small bands of Chippewa Indians.

INORGANIC STANDARDS FOR THE COLORIMETRIC CARBON TEST.

BY THEODORE W. ROBINSON, JOLIET, ILL.

(Duluth Meeting, July, 1887.)

WHEREVER the amount of work renders it practicable the plan of using permanent standard solutions, in connection with the colorimetric carbon test, affords such manifold advantages that it is to be strongly recommended. That it has not attained a wider and more general application, is mainly due to the fact that, as the method is generally employed, the difficulty attending the production of the colors, and their doubtful stability when produced, have been only too evident to those who have undertaken their composition. The old method of using standard solutions made from sugar, coffee, etc., has always been hampered by the tendency of the colors to change upon exposure to the light, and the consequent liability to serious error, unless they are closely watched. Appreciating these difficulties, while recognizing the desirability of permanent colors, Mr. Magnus Troilius, at the suggestion of Prof. F. L. Erkmann, investigated the properties of cobalt, copper and ferric chlorides, with the view of determining their efficacy in the production of standard solutions. His results, as obtained, seem to have been satisfactory, but his treatment failed to eliminate many of the objections that have rendered the use of organic colors undesirable. The

following mode of procedure in the making-up of inorganic standards I have found most efficient, in point of accuracy, ease and celerity of production, and stability.

Day Standard Colors.

I will use as a range, in illustration, the carbon-contents included between the extremes, 0.50 per cent. and 0.30 per cent., as representative of Bessemer rail-steel—that branch of the industry in which the uniformity and continuity of the product renders the use of permanent standard solutions specially advisable. The principles formulated below will evidently apply to either a higher or lower carbon-range.

The salts employed are the same as those used by Mr. Troilius, viz.: the neutral chlorides of cobalt ($CoCl_2$), copper ($CuCl_2$) and iron (Fe_2Cl_6). Solutions of these are made as follows:

Cobalt Chloride.—Dissolve in water containing 1 per cent. of free hydrochloric acid (1.12 sp. gr.) in the ratio of 1 grm. salt to 1 c.c. water.

Copper Chloride.—Dissolve in water containing 1 per cent. hydrochloric acid (1.12 sp. gr.) in the ratio of 1 grm. of the salt to $1\frac{1}{2}$ c.c. water.

Ferric Chloride.—Dissolve in water containing 2 per cent. hydrochloric acid (1.12 sp. gr.) in the ratio of 1 grm. salt to 1 c.c. water.

The solution in all cases is best effected by the application of a gentle heat; and filtering through a ribbed filter assures freedom from turbidity. Free acid is used to prevent the otherwise enhanced liability to decomposition of the chlorides.

Should any difficulty be encountered in obtaining a clear solution of the ferric chloride, subsidence of the liquid for a few hours, and then decantation, will obviate the trouble.

Our purpose is now to make up a set of colors that shall correspond to those made by the nitric acid solution of steels containing in carbon, every alternate point between the extremes of 0.50 and 0.30 per cent. It is clearly impracticable to have steel standards with this gradation of carbon, while, on the other hand, to make a single color comparable with some one steel, and then by dilution make up others corresponding to all the desired percentages is liable to introduce an error.

I have two steel bars that are kept as standards, the carbon-composition of which, as shown by repeated combustions, is 0.501 per

cent. and 0.304 per cent. respectively. As the limit of error in the colorimetric test can be considered as about 0.01 per cent. for this grade of steel, I am safe in calling the first a 0.50 and the second a 0.30 per cent. metal.

From the 0.50 and 0.30 standard steels are now made up nitric acid solutions under the same conditions, as regards amount of steel taken, acid used, time of heating and temperature, as those to which the regular tests are to be subjected. I am in the habit of dissolving 0.5 grm. of drillings in 12 c.c. nitric acid (1.20 sp. gr.) and subjecting it in a steam bath to a temperature of 98°—100° C. for fifteen minutes.

The next step is to make up from the chlorides a color which shall exactly match the 0.50 per cent. steel solution, and another which, diluted to $\frac{3}{5}$ of its strength, shall precisely correspond to the 0.30 per cent. steel. The former we will call a brown 0.50 color, and the latter a green 0.50 color. I use the term "green 0.50" for the latter, because it will be found to be a color which, though of a greener cast, corresponds in *intensity*, before diluting, to the solution of the 0.50 steel.

These are prepared as follows:

For the brown 0.50 color, mix the chloride solutions and water containing 0.8 per cent. (1.12 sp. gr.) hydrochloric acid in the ratio of 87.5 c.c. water, 5.7 c.c. cobalt chloride, 2.5 c.c. copper chloride and 4.3 c.c. ferric chloride.

For the green 0.50 color, take 82.6 c.c. water containing 0.8 per cent hydrochloric (1.12 sp. gr.), 5.4 c.c. cobalt, 5 c.c. copper and 7 c.c. ferric chloride solutions. The first of the colors so produced will approximately match the 0.50 steel solution, while the second, diluted to $\frac{3}{5}$ of its strength with water containing 0.8 per cent. hydrochloric acid (1.12 sp. gr.), will approximately correspond to the 0.30 steel solution. The absolute identity of the colors may be obtained by the addition of the chloride or chlorides, the color of which corresponds to the shade that is wanting. The comparison of the green 0.50 color with the 0.30 steel is readily accomplished by the aid of a reading-tube suitably calibrated, 6 c.c. and 10 c.c., for instance. In the final matching, care should be exercised that the same light is used as is to be regularly worked with—the same window, if possible—as this assures similarity of conditions as regards transmitted and reflected light.

With correct brown and green 0.50 colors so obtained, the alternate intermediate points are easily acquired by mixing the two and

diluting with water containing .8 per cent. hydrochloric acid (1.12 sp. gr.) by the aid of a burette, as per following table:

Per cent. Carbon.	No. c.c. Brown, 0.50 Color.	No. c.c. Green, 0.50 Color.	No. c.c. Water.
0.50	10.00	0.00	0.00
0.48	8.64	0.96	0.40
0.46	7.36	1.84	0.80
0.44	6.16	2.64	1.20
0.42	5.04	3.36	1.60
0.40	4.00	4.00	2.00
0.38	3.04	4.56	2.40
0.36	2.16	5.04	2.80
0.34	1.36	5.44	3.20
0.32	0.64	5.76	3.60
0.30	0.00	6.00	4.00

This table is based on the following considerations:

1. That theoretically, from a given solution representing carbon of known value, other solutions representing definite lower carbon percentages may be made by diluting with water.

2. That practically this will give colors of shades at variance with those aimed at.

The explanation is clear. We have first to decide upon some definite bulk as a suitable amount of the solutions to make up. With the $5'' \times \frac{5}{8}''$ reading-tubes adopted here, 10 c.c. has been found convenient. To make the 0.48 standard, for instance, we must have such a 0.50 color that the shade produced upon diluting with the proper amount of water will match that made from a 0.48 steel. This neither the green nor the brown 0.50 alone will do. The former is too green, the latter too brown.

The operation of determining in the mixture the correct proportion of the two requisite to give the desired color (for the 0.48 color as an example) is purely a mathematical one. The computation is as follows:

Amount of green : amount of brown : : 0.02 : 0.18; 0.02 and 0.18 being the relative distances of the 0.48 color from the extremes, 0.50 and 0.30. The proper ratio of the brown and green 0.50 colors being found, the amount of the mixed chlorides and water to make a total of 10 c.c. is easily calculated thus:

$$0.50 : 0.48 : : 10 : x$$
$$x = 9.6$$
$$10 - 9.6 = 0.40$$

Hence 9.6 will be the number of cubic centimeters of the combined chloride solutions and 0.40 the number of water.

Any of the other gradations in this or any range is found in a like manner. If it is desirable to make either a higher or lower, all that is necessary is to accurately know the carbon-contents of two suitable steels and proceed as above.

It is always well to keep the colors from exposure to the light as much as possible. When it is necessary to use the colors constantly, as in testing every heat at a Bessemer plant, a reading-rack capable of being closed is to be recommended. This avoids the necessity of removing to a dark place. Such a rack, provided with a front and back door, swung on hinges fastened to the bottom, and arranged with a thin plate of fine ground glass for a reading-background, has proved very effective. For this purpose I have found the glass to possess advantages over the mediums ordinarily employed. The solutions must of course be corked tight.

With due regard to the precautions and directions here laid down, a set of reliable inorganic standard solutions can be made from the above chlorides that will be found to be constant for four or five months of continued use, and often for even a longer period. When decomposition does set in, the colors gradually grow dark.

Night Standard Colors.

As the method is commonly conducted, standards are weighed from the steel direct. When tests have to be made after every blow, this is obviously inconvenient and renders it unadvisable to leave the work in unskilled hands.

With the purpose of placing the night-tests as nearly as possible on an equal footing with the day-tests, the following investigation was undertaken by the writer.

The day chloride standards, when compared by both kerosene and gas light, were found to appear much lighter than the corresponding steel solutions. The arc light gave the same result. Monochromatic light as obtained by the soda bead and various colored glasses was tried without success. The magnesium light gave the same reactions as daylight, but was laid aside as impracticable, and the idea of using the day standards for night work was then abandoned.

Night-standards that proved entirely satisfactory were finally made up from the same solutions and on exactly the same principle as the day-standards, with the simple modification that the brown

0.50 and green 0.50 were matched with the 0 50 and 0.30 steel solutions in a dark room, by the aid of kerosene light.

To make the brown 0.50 night-standard, mix the solutions in the following ratio: 7.4 c.c. cobalt, 1.5 c.c. copper and 4.3 c.c. ferric chloride solution with 86.8 c.c. water containing 0.8 per cent. hydrochloric acid (1.12 sp. gr.).

For the green 0.50 night-standard, mix in the ratio of 7.7 c.c. cobalt, 2.2 c.c. copper and 4.5 c.c. ferric chloride solution with 85.6 c.c. water containing 0.8 per cent. hydrochloric acid (1.12 sp. gr.).

The extremes being made up, the intermediate colors are produced by the same table as was used for the day-standards. The color and intensity of the brown and green 0.50 thus prepared will be approximately correct. The final shading must be accomplished in the same manner as for the day-solutions. Colors so produced will be found to be considerably darker than the corresponding day-colors, when compared by day or artificial light.

For this work and for the regular night-tests I use a rectangular camera, made of sheet-iron, arranged on the back with a door, open in front and so cut, about midway between the ends, as to admit a light wooden rack holding the colors.

Three medium-sized kerosene lamps, placed on a stand in front, furnish the light. The rack is provided with a plate of ground glass as a reading background, and is so adjusted that a slide between the glass and the lamps protects the standards from the light, when not in use, while the door on the back of the camera serves to darken the other end. With standards made as above, I have found no difficulty in turning out as accurate work by night as by day, and with the requisition of no greater skill and ability than is required of any day carbon-boy. In these night-standards, it will be noticed that there does not exist that sharp contrast in shade between the 0.30 colors produced by diluting the brown and the green .50 as is found in the day-standards. It is almost entirely a case of comparative intensity, when, unlike the day-standards, the shade is of but little moment.

The kerosene light seems to eliminate to a great degree the power of distinguishing variations in the green tint; and for this reason a very good set of night-colors could be made by simply diluting the brown extreme with water alone. It will not be as accurate a match, however, as if the green is used also.

It takes a little time to familiarize the eye to night-reading; but when once accustomed it has been my experience that as close, if

not closer, discriminations can be made between the colors by night as by day.

In comparing by dilution, the failure to take into consideration the distinction of shade as well as intensity has often been the source of considerable error. With standard solutions made as directed, this liability is reduced to a minimum, and this, with the manifest advantages of ease and celerity, renders inorganic standards a desideratum for any laboratory dealing with continuous work of this kind.

A CRYSTALLINE SUB-SULPHIDE OF IRON AND NICKEL.

BY J. B. MACKINTOSH, LEHIGH UNIVERSITY, SOUTH BETHLEHEM, PA.

(Duluth Meeting, July, 1887.)

SOME months ago I received a sample of a highly crystalline product occurring in the hearth of the shaft-furnace used in smelting the roasted niccoliferous pyrrhotite at Mr. Joseph Wharton's works near the Gap Mine, Lancaster County, Pa.

Mr. Wharton first observed this product in cavities in the concretions in these furnaces about the year 1870, and published a description of it in that year.* Thinking that it might show some analogy to meteoric iron, he sent the present sample to Mr. George F. Kunz, who turned it over to me, with the request that I should analyze it at my convenience. The results do not show the expected analogy, because the sulphur in meteoric iron is combined as FeS., while in this product there is only one-sixth the necessary amount of sulphur, which, however, seems to be combined with all the metallic elements present, forming a sub-sulphide of definite composition. I am informed by Mr. Wharton that he has had for a long time the sample which I have analyzed, and that it probably is about the same age as that which he first described. It is worthy of note that the material does not show any signs of oxidation, but that the crystals preserve a brilliant metallic lustre.

The first two analyses, made by Mr. J. Voigt, are those given by

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* *American Journal of Science*, vol. 49, p. 365, On Two Peculiar Products in the Nickel Manufacture.

Mr. Wharton in his paper above referred to; the third is the analysis I have just completed.

	Voigt. Octahedral crystals.	Granular mass.	Mackintosh. Cubical crystals.
Cu,	1.85	1.74	2.20
Ni, Co,	25.22	28.20	26.16
Fe,	64.10	62.50	61.685
S,	8.90	7.60	8.305
SiO_2,	.....	.....	0.56
	100.07	100.04	98.91

Theoretical for Fe_4Ni_2S.

Ni,	31.29
Fe,	60.12
S,	8.59
	100.00

In my analysis there is a small amount of silica which is present in mechanical admixture, in the form of slag or furnace-lining. The deficiency in the analysis is due to the other constituents of this admixture. In the granular mass, the sulphur is deficient; but as it *was* a granular mass, on which the octahedral crystals were imbedded, it could not be expected to be a definite compound. In the case of the other two analyses, however, both made on crystalline material of approximate purity, the results are quite accordant and agree with a compound of the general formula R_6S. The ratio of the metals is, roughly, two of iron to one of nickel and copper, and it will be seen, from the theoretical composition which I have calculated on that assumption, that the results of analysis do not vary very much from those calculated.

From a chemical standpoint this compound is of great interest, as it distinctly illustrates the hexatomic character of sulphur, and to the best of my belief is the only known sulphide of similar composition. The manner of occurrence is in small cubical crystals, forming fern-like aggregations, in the cavities in the concretions in the furnace-hearth.

I think it is quite probable that analogous compounds will be formed in copper-smelting; and I would be pleased to receive samples of any similar product from any one whose attention may be directed to this note.

THE CHAPIN IRON-MINE, LAKE SUPERIOR.

BY PER LARSSON, IRON MOUNTAIN, MICH.

(Duluth Meeting, July, 1887.)

THE Chapin Mine, on the Menominee range, Lake Superior, was first opened in 1880 and has since then produced 1½ million tons of soft blue hematite, containing about 63 per cent. of iron and 0.07 per cent. of phosphorus.

Up to the present time, the ore has been mined from three separate lenticular deposits, which conform with the Huronian strata of the district, striking N. 75° W., dipping from 70° to 80° N., and pitching about 30° W.

The largest of these ore-bodies has a horizontal length of 2500 feet, and a maximum width of 130 feet in the middle, from which it gradually narrows to the pointed ends. Its dimensions have not changed materially as greater depth has been attained. Figs. 1 and 2 show the geology and the mine-workings on the main deposit.

As this main deposit pitches away from the eastern part of the property, it is replaced by another lens, containing ore of the same quality. When first opened on the 1st level, this lens was 150 feet long and up to 50 feet wide, while now on the 6th level it shows a length of 550 feet, and a maximum width of 100 feet. It is separated from the extreme east end of the main deposit by about 30 feet of clay-slates. It pinches out gradually between the slates to the west, and stops abruptly against the jasper to the east.

Farther east in this belt of jasper occurs the third ore-lens which has a pretty regular length of 100 feet and averages, from the 1st to the 5th level, 20 feet in width.

A cross-section of the ore-formation shows, on the north or hanging side of the ore, about 200 feet of clay-slates, and farther north a heavy belt of magnesian limestone. The slates and the dolomite are generally separated near the surface by a conglomerate of broken dolomite and soft slates.

The foot-wall rock is also a clay-slate, containing a higher percentage of iron and less magnesia than the hanging-slate. Farther south occur alternate beds of slates and lean ore or jasper.

Methods of Mining.—The first work done in the Chapin mine was on the extreme east end of the main deposit. As the ore at this point was of only moderate width, it was first mined by "stull-stoping," a method quite common in the Lake Superior hard-ore mines. In this way of mining, heavy pillars of ore are left around the shafts and large rooms are made in the ore by back-stoping, after which, in order to keep up the hanging-wall, rows of long heavy timbers are placed across the excavated rooms from foot- to hanging-wall, and covered by light poles or lagging to prevent pieces of ore that may drop from the roof from falling on the men.

It was, however, very soon found that this way of mining was impracticable, both on account of the great width of the ore-deposit and, also, on account of the soft nature of the surrounding rock and of the ore itself.

Another method was then adopted, which was called "the modified Nevada system." The shafts were still kept in the ore and protected by heavy shaft-pillars, and the excavations were made secure by timber, though in a different manner. After the shafts had been connected by levels driven near the foot-wall, and these levels timbered with drift-sets, rooms were laid out 18 feet apart, and the ore was worked out towards the hanging-wall by breast-stoping far enough to make room for one or more sets of room-timbers. The first row of sets was put on heavy mudsills of flat timber, laid across the room 8 feet apart from center to center, and extending two feet into hitches cut in each of the adjoining room-pillars. To these mudsills were spiked two parallel stringers of lighter flat timber, which were placed near the sides of the pillars and thus across the mudsills, and also across the trend of the ore. In these mudsills dovetail-mortises were framed to receive the tenons of the upright legs or posts. Tenons were also framed at the other ends of the legs to fit into mortises in the 18 feet long caps of 16″ x 18″ timber, which were laid across the room from leg to leg. In this way room-sets were placed 8 feet apart from foot-wall to hanging-wall and finally lagged over. Lagging had also been placed between the legs and the sides of the pillars in order to keep the ore from crumbling. Ore-chutes were also built between alternate sets, to receive the ore from the upper stopes.

A second stope, on top of the timber-sets, was started from a winze, which was sunk both for ventilation and for the purpose of letting down timber from the level above. As the second stope was

mined, a row of room-sets similar to that just described was placed above the first, having their legs mortised into stringers spiked to the ends of the caps. In this way the whole lift was mined.

It is obvious that care had to be taken in mining out the last stope of a lift, that the mudsills in the room above should be properly taken up. This was not difficult, as the bearings of these mudsills in their hitches, cut out in the room-pillars, were sufficient to hold them up temporarily.

In this manner 750,000 tons of ore were mined at a reasonable cost very rapidly.

In the latter part of 1884, when mining had been carried on under this system for three years, the room-pillars in the west—which is the widest—part of the mine commenced to show signs of crushing, and it became apparent that it would not be safe to open new rooms at a greater depth.

It may be said that the pillars were not laid out wide enough; but it should be remembered that fully one-half of the ore was left in the mine, which fact in itself was a great objection to the rooming method. An attempt was, however, made to increase the size both of rooms and pillars, but it was unsuccessful on account of the soft nature of the ore.

It was then decided to abandon the so-called "modified Nevada system," and adopt one or two of the old standard filling-systems that had been successfully employed in similar European mines for so great a number of years. One consideration that helped to bring about this change was the fact that the supply of such heavy timber as was required for room-sets was rapidly becoming exhausted in the neighborhood of the mine.

Shortly after this change had been decided on, an extensive cave occurred between No. 9 and No. 10 shafts, whereby three room-pillars, containing more than 100,000 tons of ore, were crushed and more or less mixed with the surface-sand. This cave showed conclusively that the rooming had not been abandoned any too soon, and that its abandonment was a matter of absolute necessity.

It is, however, not at all desirable *per se* to change the method of working in a mine—a fact well known to every mining man. In this case it caused a very material falling off in the production, and necessitated many expensive changes both on the surface and underground. New shafts had to be sunk from the top, and new arrangements made for handling the ore.

Two new shafts, "B" and "C," were started 300 feet from the ore in the solid dolomite on the hanging-wall side, and cross-cuts were driven from the mine to meet these shafts.

In order to keep the standing room-pillars in place, and thereby prevent new caves, the drift on top of the ore was excavated and dumped into the rooms through small shafts sunk into the "back" of each room. This work was successfully completed except in the west end of the mine, where three room-pillars gave way and let in the roof. Ore is now being drawn from the bottom of these broken pillars; but it is evident that nearly one-half will be lost. The other pillars are now being mined under the filling-system, but though the work at present is progressing favorably, it is yet too early to determine how nearly the full amount of ore left in them can be recovered.

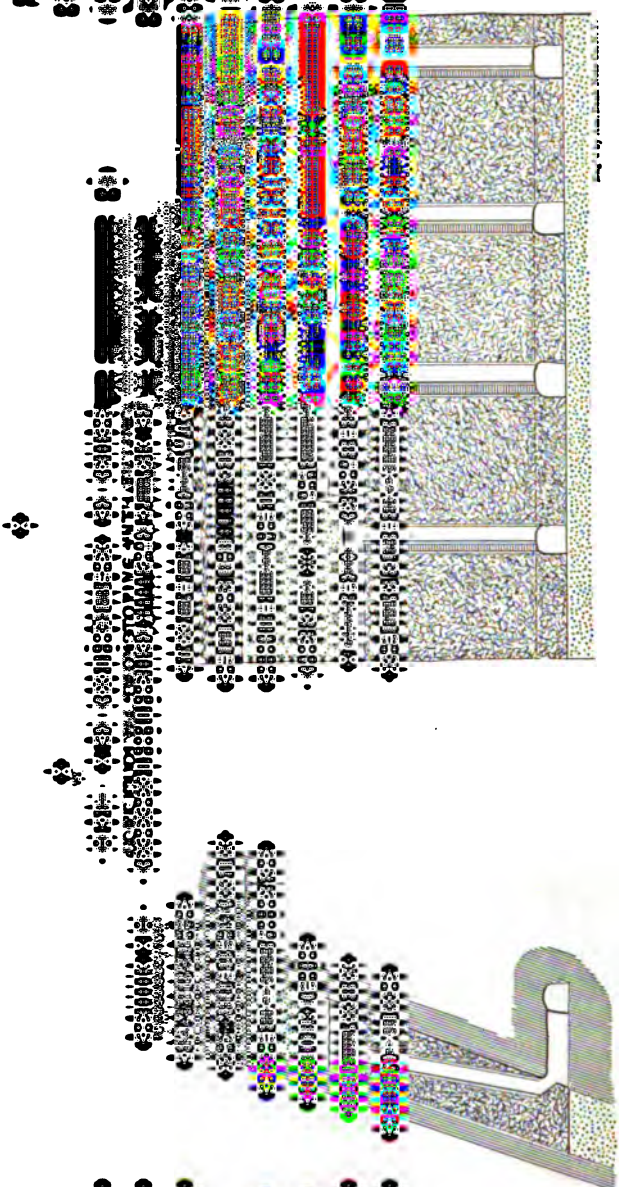
In opening new ground, two of the different methods of mining, where rock-filling takes the place of the ore, have been employed, viz.: *longitudinal back-stoping with filling* (the *Firstenbau* of the Germans) and *transverse back-stoping with filling* (*Querbau*). Fig. 4 illustrates the first of these methods, and Fig. 3 the second.

The work preparatory to stoping is about the same in both these methods. From the shafts, sunk in the hanging- or foot-wall, cross-cuts are driven to the ore and connected by drifts running some 20 or 30 feet from the ore, either in the hanging- or foot-wall rock. Owing to the regularity of the Chapin ore-deposits, these drifts can be run perfectly straight for long distances, at a nearly uniform distance from the ore. In regard to the location of these main drifts, it has been found necessary in the Chapin to place them on the foot-wall side of the ore. An attempt to locate them in the hanging-wall failed. The rock that takes the place of the ore yields somewhat under the weight of the hanging-wall, whereby heavy layers of the latter are broken off from soapy slips running with the stratification. No timber can be put in strong enough to resist the pressure caused by this movement of the ground. The foot-wall drifts have not as yet shown any signs of crushing.

At the time the main levels are driven, smaller drifts are run in the ore, and from time to time connected with the rock-drifts for ventilation.

Longitudinal Stoping.—If the ore is firm and not over 20 feet wide, which seldom happens, the full width is mined lengthwise on the trend of the deposit. As the stope advances, filling is trammed

of the rock-winze in small
e stope. In this way, shovel-



NOTE.—For explanation of markings, see Fig. 3.

, the filling must be kept at
ough to allow the trammers to

Transverse Stoping.—Where the ore is wider than 20 feet, the filling must be kept close to the back. The miners then proceed in the following manner: On the first stope different parties commence to work 50 feet from each other in the ore-drift run parallel with the main level, and make cuts about 8 feet high and 9 feet wide clear across the ore. If the ground is weak, props or sets of light timber are put up as the cuts advance.

These openings are then filled with rock, either before or at the same time as other cuts of the same size are made, alongside of the first ones. A third slice is then taken off, and the second is filled in the same manner; and so on, until the whole first stope is mined out. As the filling must be kept close to the back, in order to prevent caving of the ore, it is necessary to shovel most of it. It should, however, be borne in mind, that as solid ore is mined and loose rock takes its place, and as the specific gravity of the ore is at least $1\frac{1}{2}$ times greater than that of the rock, it is not necessary to handle more than four tons of rock for every ten tons of ore mined. As soon as the filling is put in, it is planked over.

Before work is commenced on the second stope, ore-chutes and rock-winzes must be prepared. Raises to be used for ore-chutes and ladder-ways are made from the side of the main level to the top of the second stope, and cross-cuts are driven into the ore. These chutes could be located in the ore and connected with the main levels by cross-cuts; but as the tramming will shortly be done by machinery, it is preferable to have the chutes open directly into the levels. The ore-chutes are placed 50 feet from one another, and the rock-winzes are sunk 100 feet apart from the next higher level.

The first thing to do on the second stope is to connect the rock-winze with the cross-cut leading to the ore-chute, after which the ore will be taken out and filling brought in in the manner above described.

A third stope is then prepared and mined in a similar way, and so on until the whole lift is mined out.

On account of the soft character and the great width of the ore, it sometimes cracks off and settles down on the filling. This will not cause much difficulty, if the filling is kept up close to the back on every stope.

If a block of loose ore is met with, it is necessary to put up drift-sets and drive laths, in order to keep the ore from running.

The ore-chutes are cribbed up for the first 20 feet large enough to hold about 50 tons of ore, and then narrowed up to a size of 2½ feet square. From this point they are built circular, 2½ feet in diameter, by means of wedge-shaped blocks of wood cut out in the saw-mill. Care is taken in dumping the ore into the chute that it is not allowed to accumulate and rise in this circular part. Ladder-ways are cribbed up on the side of each ore-chute.

The rock mined in drifts and shafts is, of course, used to fill the excavations in the ore. If, however, this rock is not sufficient, a very suitable filling-material can be obtained from a sandstone quarry a few hundred feet from the mine. The sandstone is trammed to one of the hoisting-shafts, and lowered, on one of the balanced cages used for hoisting, to the level next above the lift where the ore is mined. From the shaft it is trammed on this level to one of the rock-winzes and dumped. It is then drawn from a temporary chute built at the bottom of this winze, and trammed to its destination.

Up to the present time the ore taken from the ore-chutes has been trammed in one-ton cars by men. Hereafter two-ton cars will be used, and the power will be taken from a pair of 12″ x 36″ horizontal Corliss engines placed near C shaft on the 6th level. These engines will be run by compressed air, and will drive two endless ropes, one for the 5th and the other for the 6th level. The speed of the rope will be about 1½ miles per hour.

The levels are provided with two tracks, one for empties and the other for the loaded cars. The empty cars will be coupled to the rope as fast as they come down from the surface, and taken to the end of the track. Here a man will be stationed to move them over to the ore-track. The ore-fillers will then stop them at the chutes from which the ore at the time is being drawn, and couple them to the rope as soon as they are loaded.

The two main hoisting-shafts B and C are sunk vertically 9′ x 12′ inside the timber, and equipped with two balanced cages.

The hoisting-engines, which are placed in stone houses 110 feet from the shafts, have 30″ x 60″ double cylinders acting direct on two conical wrought-iron drums of an average diameter of 12½ feet, provided with friction driving-bands and safety-brakes.

Considerable ore is also at present raised through the pump-shaft A, and another underlay-shaft, sunk lately in the foot-wall near the east end of the mine, for the purpose of hoisting the ore from No. 3 and No. 5 shaft-pillars and adjoining room-pillars.

The distance from A to B shaft is 750 feet, and from B to C, 775 feet. About 1100 feet west of C and 600 feet east of the western boundary line of the Chapin property, a new combined pumping and hoisting-shaft is just being started in the foot-wall. As it will have to be sunk through 98 feet of water-bearing sand and gravel, it has been decided to make use of the Poetsch freezing process. It is thought that this system of sinking shafts will not only make it possible to reach the ledge in a shorter length of time, but will also leave the shaft and the ground around it in a better condition than if any other method were employed. The shaft will be 22 feet in diameter, and lined with cast-iron tubing down to the ledge.

During the shipping season, the ore is trammed from the hoisting-shafts by machinery and dumped into pockets, from which the railroad-cars are filled.

During the winter months it is dumped from a trestle 35 feet high on an ore-dock, which is 1200 feet long and 130 feet wide, and is elevated about 9 feet above the railroad-track.

Most of the mine-water is raised 500 feet from the bottom of A shaft by two sets of Cornish pressure-pumps. The geared pumping-engine at A shaft has two horizontal non-condensing 24″ x 48″ cylinders. It works two sets of plungers, 12″ and 17″ diameter, raising about 1200 gallons per minute.

The mining machinery at the Chapin, and also at its neighbor, the Ludington mine, is driven chiefly by compressed air furnished by a plant of air-compressors located at Quinnesec Falls on the Menominee river, about 3 miles from the mines.

This plant consists of three pairs of 32″ x 60″ compressors, driven by three independent vertical 48″ inward-flow turbines, and one pair 36″ x 60″ compressors, driven by a 54″ turbine of the same type. The power given out by the turbines is transmitted to the compressor-pistons by gearing, the speed being reduced about four times.

The compressors, which were built by the Rand Drill Company, of New York, have water-jacketed cylinders and heads, and hollow pistons for water-circulation. Each pair of compressors has independent foundations, the loading being 10 feet high. The spaces between the foundations form tail-races for the turbines placed at their upper end.

The water is led to the turbines through iron penstocks 7′ to 8′

in diameter from a flume starting a short distance above the fall, the height of which is 47 feet.

The average delivery of air per 24 hours during last year was 1,827,350 cubic feet of 60 lbs. pressure and 60° F. temperature. The average speed of the three pairs then running, if time of stoppages be not included, was 29.41 revolutions per minute, and the average pressure in the receiver was 60.84 lbs. The pressure at the mines was from 2 to 3 lbs. less.

In conclusion, the following tables, showing the effect of labor and powder at the Chapin mine, are presented:

Effect of Labor.

Year.	Tons per man per day.			Feet per man per day.		
	Broken in the stopes.	Loaded and trammed in the mine.	Loaded and trammed from st'kpile.	Drifts.	Winzes.	Shafts.
1884	9.46	23.75	24.4	.513	.479	.077
1885	6.37	20.08	25.4	.535	.774	.081
1886	6.77	16.40	31.9	.610	.498	.115

Effect of Powder* (in contract work).

Year.	Number of tons broken per lb. powder.	Drifts.			Winzes.			Shafts.		
		Feet driven.	Lbs. powder used.	Lbs. powder per ft.	Feet sunk.	Lbs. powder used.	Lbs. powder per ft.	Feet sunk.	Lbs. powder used.	Lbs. powder per ft.
1884	5.96	4024	11689	2.90	2208	4879	2.21	620	5320	8.58
1886	5.00	10159	29113	2.86	1938	4235	2.18	242	2455	10.13

* "Atlantic," with 27 per cent. of nitroglycerine.

TRANSFERRED TO GEOLOGICAL SURVEY

GEOLOGICAL MAP
of
CHAPIN MINE

LONGITUDINAL

THE CANADIAN IRON TRADE.

BY JAMES HERBERT BARTLETT, MONTREAL, CANADA.

(Duluth Meeting, July, 1887.)

I DESIRE, very briefly, to call the attention of the Institute to the iron trade of the Dominion of Canada.

A wonderful development is now going on in Canada. The completion of the Canadian Pacific Railway, and the numerous extensions of the various railways all over the country, combined with the progress made in nearly every kind of manufacture, is drawing attention to the field offered in Canada for the profitable employment of capital, and specially to the opportunity there is for the manufacture of iron.

In a paper presented to the Institute at the Halifax meeting,* the writer gave a review of the various attempts to manufacture iron in Canada. The facts then presented were not such as to offer much encouragement to any one to embark in similar enterprises under the conditions then existing, but, happily, these conditions are now changed, and an early development may be anticipated.

There are two reasons which make the present time an appropriate one to direct attention to the field offered for enterprise in this direction, the first being the very favorable reports of the experts who examined and reported on the exhibit of minerals at the Colonial and Indian exhibition, held in London, England, last year. The exhibits there of coal and iron attracted so much attention that at the request of the Iron and Steel Institute a paper on the "Iron-making resources of our Colonies as illustrated by the Colonial and Indian Exhibition," was prepared and read at the last meeting of the Institute.† This report of 135 pages is very exhaustive, and the conclusion arrived at, so far as Canada is concerned, is, that she has in many parts of the country every natural facility for the manufacture of iron.

The second and more important reason is the recent change in the Canadian customs tariff inaugurated in May last. The tariff is now generally two-thirds of the American tariff, in addition to which the Government, in order to encourage the manufacture of iron, grants a bounty upon pig-iron made in Canada out of Canadian ore.

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* Vol. xiv., p. 508.

† *Journal of the Iron and Steel Institute*, 1886, vol. ii., pp. 546–614.

The present tariff changes in full are to be found in the Appendix.

It will be remembered that the Provinces forming the Dominion of Canada were only confederated in the year 1867. Previously they were all separate and distinct colonies, each with its own fiscal tariff. After confederation the customs tariff was simply a revenue not a protective tariff, until the year 1879, when an import duty of $2 per ton was imposed upon pig-iron which previously had always been admitted free of duty; but the iron section of the tariff was still very incomplete. As Canadian iron-workers' wages are regulated by Pittsburgh wages, it is impossible, unless sufficient protection is afforded, to compete with Belgian, German and English manufacturers whose wages are on a much lower scale.

With one single exception all manufactures of iron in Canada have been made with charcoal as a fuel. Many of the attempts were made early in the century, when the operations were of very small dimensions. The only furnaces which have been successful have used bog iron-ore and have made only a few tons of iron per day.

The various attempts to make charcoal pig-iron and blooms in the four older Provinces of Ontario, Quebec, New Brunswick and Nova Scotia, have generally met with a want of success. The failures are clearly attributable to want of protection, the sparsely settled condition of the country, want of transportation facilities, and of judgment in the selection of the situation.

The market, as may be supposed, was at the time very limited, as, saving the small quantity of charcoal pig-iron required in making car-wheels, there was no home market, except in competition with imported coke-iron, to meet which was impossible. A variety of experiments have been made, which are counted as failures in the iron manufacture, bringing unwarranted discredit upon this industry. The experiments in many cases were costly, and were undertaken by patentees of all kinds, of no standing or experience in the trade. The only attempt ever made to manufacture coke pig-iron and refined bar-iron has proved the possibility of doing so. The quality of the article is in both cases superior to that imported and commands a higher price.

The importance of the manufacture of iron to the country generally will be appreciated when it is stated that the total balance of trade against Canada from 1868 to 1886 was $381,000,000. The total value of the imports of iron and steel during that period was $253,250,000, from which it may be assumed that the balance of trade would have been in our favor, had we made our own iron.

A great bar to the trade has been a want of information regarding the extent of the market to be supplied. This difficulty has been to some extent removed, the writer having collected and tabulated the statistics of the Canadian iron trade for all the years since confederation, so that it is now possible to see what has been required in the past, and to estimate what will be necessary in the future.

Some particulars of the nature and extent of the Canadian iron trade may be of service at the present time as a guide to any one desirous of ascertaining facts necessary for the direction of enterprise in this special field. The confederation of the Dominion of Canada took place in 1867, and particulars of the trade only date from then.

The following table and the diagrams will show the Canadian iron trade.

Imports of Iron and Steel and Manufactures thereof into the Dominion for Home Consumption for years.

1868,	$6,885,365
1869,	7,385,780
1870,	7,750,867
1871,	10,808,645
1872,	15,913,179
1873,	25,435,020
1874,	20,700,387
1875,	18,199,198
1876,	12,965,117
1877,	11,082,321
1878,	9,398,306
1879,	7,962,295
1880,	10,128,660
1881,	12,955,855
1882,	17,499,488
1883,	20,080,274
1884,	14,790,727
1885,	11,415,713
1886,	11,053,365
Total,	$253,210,512

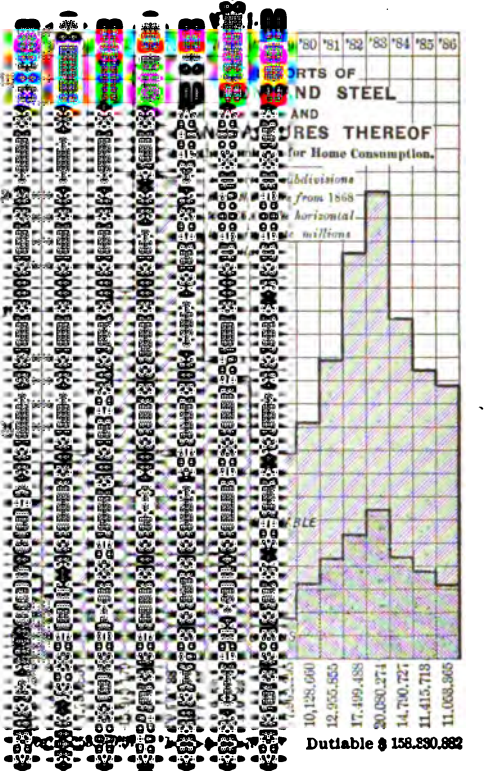
This large amount of $253,000,000 was paid out in hard cash by the people of Canada to support the iron industries of other countries. Of this amount $94,879,630 was free, and $158,330,882 dutiable.

The value of iron and steel and their manufactures imported into Canada for home consumption during sixteen years, 1868 to 1884, is over $230,000,000. The trade is divided into seven headings, the value of imports under each from 1868 to 1884 being as follows:

TRADE.

	$75,179,153
	9,938,614
	48,068,618
	9,703,717
	10,742,331
	47,926,637
	29,182,414
Total,	$230,741,484

one of iron, steel and rails
id in 1884 to 273,967 tons;
veraged over 250,000 tons.



considerable proportion of this
Pacific Railway; but apart
e have been on the average
year for the past ten years.
opulation, the amount of iron
rkable. In the year 1878 a

[illegible] ported into Canada
[illegible] iron of our own,
[illegible] *per capita* of the
[illegible] value of the im-
[illegible] seen by the fol-
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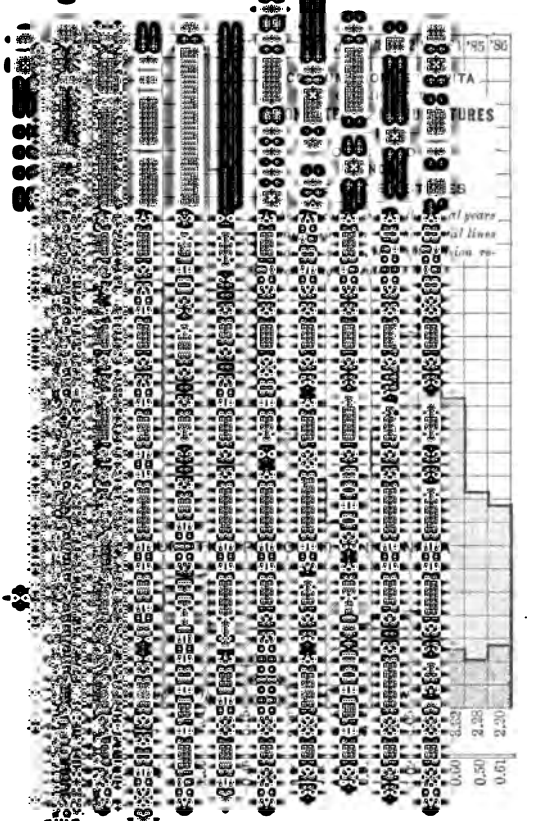


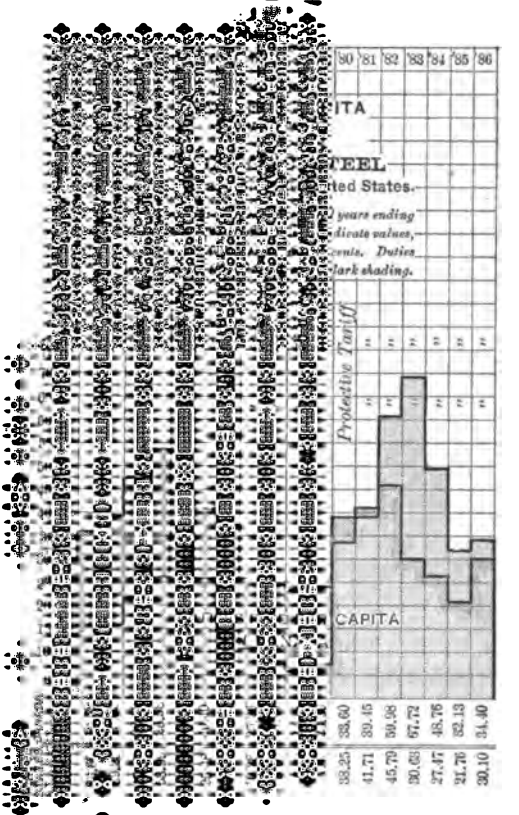
[illegible] the United States

	Canada.	United States.
[illegible]	04	$0.60
[illegible]	17	0.74
[illegible]	25	0.84
[illegible]	01	1.09
[illegible]	55	1.30
[illegible]	01	1.39

[illegible] TRADE.

	Canada.	United States.
. .	5.77	0.81
. .	5.15	0.46
. .	3.46	0.28
. .	2.89	0.22
. .	2.40	0.18
. .	2.00	0.20
. .	2.49	0.91
. .	2.98	1.00
. .	4.05	1.02
. .	4.56	0.75
. .	3.32	0.60
. .	2.28	0.50
. .	2.20	0.61

[illegible] lation upon imports of iron



[illegible] Canada and the United States are shown on the

Canadian Iron Works.

There are in the Dominion at the present time only the following iron works:

Londonderry, N. S.—Two coke blast-furnaces and puddling-furnaces, with refined bar-mill, forge, etc.

Radnor, near Three Rivers, Que.—One charcoal blast-furnace.

Drummondville, Que.—Two charcoal blast-furnaces.

New Glasgow, N. S.—Open-hearth steel furnace and rolling-mill.

Halifax, N. S. (one); *St. John, N. B.* (three); *Montreal* (four); *Hamilton, Ont.* (two); *London, Ont.* (one)—rolling-mills for reworking scrap and puddled bars, mostly into nail plate.

New Glasgow, N. S. (one); *Hamilton, Ont.* (one)—forges for reworking scrap, mostly into car-axles.

Montreal.—Rolling-mill for making butt-welded pipe; foundry for steel castings.

Toronto, Ont.; Hamilton, Ont.—Foundries for cast-iron pipes.

In Canada, as in the United States, the greatest consumption of iron and steel is created by the construction of railways. When the change from iron to steel rails took place, there were four rolling-mills re-rolling iron rails in Canada, one in Montreal, two in Toronto, and one in Hamilton. The substitution of steel rails, however, caused the mills to suspend operations, their machinery and appliances not being suitable; the mills were dismantled, and the consequent losses have not yet been entirely forgotten. But another and a more recent disaster, namely, the commercial failure of the Steel Company of Canada, Limited, has been most disheartening, effectually dampening the enthusiasm of many believers in the possibilities of the development of our natural resources in this direction. It is hardly to be wondered at that the general Canadian public should lose faith, when an enterprise so brilliantly ushered into existence should only have lasted so short a time, and with such bad results, not only to the shareholders and others financially interested, but to the country generally. When the actual facts are known the reasons of this failure will be plainly seen. These facts are here presented simply with a view to clear the iron industry from this cloud under which it at present rests.

The Londonderry Mines.

In the forests of the Cobequid Hills, in the province of Nova Scotia, miles away from any road or settlement, a vein of iron-ore was years ago discovered. Geologists visited the locality and pro-

nounced the deposits of great extent, and a grant of land was obtained from the Government. A Catalan forge was built in 1850, and three years later a small blast-furnace was put up, charcoal in both cases was used as a fuel, the trees from the forest around being made into charcoal. A small stream ran past the iron-mine, and was made to drive the blast-engine. The iron-ore was very pure, so the pig-iron made was of superior quality. There was no home market, but it was exported to England, although the iron had to be carted to the nearest shipping-point, namely, Great Village, six miles away from the furnace, and situated at the entrance to a small tidal river on the Cobequid branch of the Bay of Fundy. Navigation on the upper end of the Bay of Fundy is dangerous; the tides, which here rise to the height of 71 feet, rush in and out with great rapidity; the river could only be entered at high tide by vessels drawing not more than 12 feet of water, and the navigation to the entrance of the river was bad. The construction of the Intercolonial Railway, which eventually was brought within three miles of the furnace (at the expense of permanently lengthening the main line by five miles, and unfavorably affecting the gradients and curvature) made the place more accessible. It was still only a little hamlet in the midst of the forest, but it was self-contained, having plenty of ore and timber for making charcoal. There was a demand for the iron, as, owing to its superiority, the English War Office, upon the recommendation of Sir William Fairbairn and others, were using it for the manufacture of ordnance. This was before the age of steel; no doubt the iron commanded a high price; and, altogether, the works were remunerative to their owners.

In 1873 the Acadia Iron Mines, as they were then called, were purchased by a company of a few English capitalists having Sir William Siemens at their head with a capital of two and a half million dollars. Their intention was to make steel directly out of iron-ore by a new patented process, invented by Sir William Siemens, and also to make coke, pig-iron, etc. This company, known as "The Steel Company of Canada," had its headquarters in England and managed the Londonderry business from there. They paid $400,000 in cash and $600,000 in paid-up stock for the Acadia iron-mines, also $40,000 for the patent rights, a total of $1,040,000.

The accidental location of the little charcoal blast-furnace seems to have given rise to the choice of the situation for the new works. It would be difficult to account for the selection in any other way. Works specially designed for the new patented process, with rota-

tors, regenerative melting furnaces with gas-producers, two blast-furnaces, branch railways to the Intercolonial Railway and the different ore-mines, houses, buildings, etc., etc., were built. The scale on which the expenditures were made can best be understood by reference to the item of cost for the manager's house, which came to $40,000.

The works were completed and got into operation, probably at a further expenditure of about $1,250,000. The new process did not seem to do very well; and after costly experiments, and repeated trials, it eventually proved here, as elsewhere, a complete failure. Hundreds, perhaps thousands, of tons of expensive machinery had to be broken up, and the melting-furnaces and gas-producers were pulled down. A second-hand rolling-mill was purchased, some puddling furnaces were built, an axle-forge with a foundry for car-wheels and general castings were added, and the products of the works was changed from steel to pig-iron, bar-iron, and castings. The place was not now self-contained, as before; charcoal was no longer the fuel used, so the trees were of little use, except for timbers in the iron-mines. What was wanted was coal and coke; and, although at the time of the purchase coal was supposed to be on the property, it has never yet been actually discovered. The coal-field of Pictou is 51 miles to the eastward, that of Cumberland 43 miles to the westward. Limestone in considerable quantities is required, and is obtained from Brookfield, 25 miles to the eastward. When, after having paid freight on all these materials, iron is made out of them, there is no outlet but by the same Intercolonial Railway, the distance by rail to Montreal being 773 miles.

This condition of affairs was bad enough; but the situation was made much worse by the fact that the company had never built any coke-ovens of its own, and that, at this time, only one colliery mined a coal suitable for coking, and also owned the only coke-ovens in the country. This colliery, consequently, supplied coke at its own price, helping materially to kill the goose which laid the golden egg. One day, an explosion took place in this particular coal-mine, set the mine on fire, and closed it. It has been closed ever since, but is now being opened again. For a time coke was not to be had, at any price; the blast-furnace had to be shut down, and the loss, from this cause alone, can be better imagined than described. After this experience, some coke-ovens were built, and to some extent this has made the company independent. Finally, a coal mine was purchased and fully equipped; but, upon practical trials, the coal

was found to be to a considerable extent unfitted for the company's uses. It was also discovered that, through an arrangement to have the iron-ore mined by contract, the contractor had made money for himself, but had permanently injured one of the iron-mines.

Instead of running a general store, from which a considerable revenue would accrue, some outsider was granted the privilege, and took advantage of it.

It is not surprising that, after all these vicissitudes, and in view, also, of the fact that until 1880 imported pig-iron was admitted free of duty into Canada, the company failed. It is more to be wondered at that the struggle was maintained so long.

In 1880 an import duty of $2 per ton was imposed, and in 1883 a bounty of $1.50 per ton of pig-iron manufactured out of Canadian ore was granted by the Dominion Government. Under these improved conditions, the company's operations were continued by the liquidators. Since the change in the tariff, a reorganization has been effected. The management is now centered in Canada, and the enterprise will no doubt be made productive to the proprietors and the country at large. Under a careful management, there can be no question that a satisfactory dividend can be earned upon the expenditure of the works as they stand.

It is manifestly unfair to condemn every projected iron-making enterprise in Canada simply because, in the past, this particular case has not succeeded. In spite of all their troubles, the pig-iron, bar-iron, and other products of the Londonderry works, have been of a very superior quality, and have always commanded the highest price in the market.

Resources of Nova Scotia and Other Provinces.

In almost every Province iron-ore is found in abundance, and the Provinces which have not coal have an abundance of timber fit for making charcoal. We possess the only deposits of coal on both the Atlantic and Pacific shores; in Vancouver's Island and Cape Breton the coal seams run out under the ocean. In both these Provinces, British Columbia and Nova Scotia, nature has been prodigal with deposits of various descriptions of iron-ore of very good quality, and with plenty of flux, all in the immediate vicinity of the coal-fields.

There are now a number of collieries mining a good coking-coal in Nova Scotia; and the railway construction now progressing there

will bring an unlimited supply of fuel within reach of all parts of the Province.

In the county of Pictou, N. S., iron, coal, and limestone are found in the closest proximity; within a radius of ten miles there is everything necessary to build up a great and successful industry; and this mineral wealth is found on the seaboard, so that the products of the industry can be cheaply transported by water to the head of Lake Superior, if required; several descriptions of iron-ore are found in this county. There is specular iron-ore, with 68.33 per cent. of metallic iron; compact limonite, with 57.71; fibrous limonite, with 59.50; red hematite, and also fossiliferous hematite, with 54.36, and spathic ore, with 43.56.

A radius of ten miles from Stellarton would include the entire Pictou coal-field, four large collieries in active operation, producing over half a million tons of coal per annum, a coal particularly suitable in quality for the manufacture of coke and for iron working. Within this radius, too, would be included immense deposits of limestone, and five different varieties of iron-ore, with deposits of fire-clay and moulding-sand. It would take in Pictou harbor, the finest harbor on the south shore of the Gulf of St. Lawrence, with five loading and shipping wharves and piers, and over 75 miles of railway, radiating from Stellarton; and would also include the towns of Pictou, New Glasgow, Westville, Stellarton, and many villages and hamlets, aggregating over fifteen thousand inhabitants, together with a steel-works and forge, foundries, boiler and machine-shops, glass-works, saw-mills, stone-quarries, brick-yards, ship-yards, etc., some of the largest ships ever built in Canada having been launched from here.

The Cape Breton coal-fields are only 180 miles distant by sea. A railway to reach them is now in course of construction; another railway is building to the westward, which will reduce the distance from the Cumberland and Joggings coal-fields to 75 miles, instead of 112 miles as at present.

The Inspector of Mines for the Province of Nova Scotia says in his last annual report: "It may be remarked that in Pictou County the conditions for making iron and steel cheaply are unsurpassed, as within a few miles are collected numerous iron-ores, fluxes, and good furnace-fuels, and there is railway and water communication with all parts of the Dominion."

In New Brunswick there is hematite iron-ore; in the county of Carleton ore and coal are to be found, and also in different districts

of that Province. Bog-ore occurs in Queen's, Sunbury, York, Charlotte, Restigouche, and Northumberland Counties, in close proximity to the Intercolonial Railway.

The iron-ores in Quebec are magnetites, occurring more or less throughout the Laurentian range of mountains along the Ottawa river. At the Hull or Baldwin mines, west of the Gatineau river, the ore analyzes 67 per cent. The Haycock mine ore ranges from 64 to 68 per cent. of metal; the Bristol mine 58 per cent. In Three Rivers and Drummondville there is bog iron-ore in abundance, and unlimited timber for making charcoal.

In the eastern district of Ontario, back of Kingston, Belleville, Trenton, and Cobourg, between Lake Ontario and the Ottawa river, in the townships of Marmora, Hastings, Tudor, Bedford, Madoc, Wollaston, Palmerston, Bagot, Belmont, Darling, Barrie, Galway, Snowden, and many others, there is magnetic iron in abundance. This district is served by the Ontario and Quebec, Kingston and Pembroke railways, the Central Ontario, Cobourg and Peterboro', and Grand Trunk railways, and by the Rideau and Trent canals. There is plenty of timber all through this district. A surveyor who has explored through these districts, and away up 130 miles further, to the intersection of the Canadian Pacific Railway, assures me that not only was there an unlimited quantity of iron through that district, but that the forests along that line of communication could not be surpassed in any country in the world for the production of charcoal. There are also deposits of iron-ore in many other parts of the Province. In Manitoba there is iron-ore, and they have about 15,000 square miles underlaid with coal.

In the Northwest Provinces, the future great wheat-producing country of the continent, there are deposits of iron-ore reported, but the country is so vast no special explorations have been made for it. They are working large deposits of anthracite coal at Anthracite, N. W. T., of a quality which compares favorably with the best from Pennsylvania, and there are 500 square miles of this section. The climate of this country is cold, but 50,000 square miles of it is underlaid with coal of good quality, easily reached and worked, in seams estimated to yield from five to nine million tons per square mile.

San Francisco is largely supplied with Nanaimo coal, and the blast-furnaces in Oregon get most of their iron-ore from Texada Island in British Columbia. The eastern cities of Portland and Boston are supplied with some of their coal from Nova Scotia, and

it would appear possible also to supply iron-ore to eastern furnaces from that Province.

Disposition of the Government.

There is at present an opening in Canada for the manufacture of iron. In the Canadian hand-book for the Colonial and Indian Exhibition, prepared under the direction of the Hon. John Carling, Minister of Agriculture, it is stated, "for a country having 11,000 miles of railway, with a weight of over a million tons of rails, and possessing for the manufacture of iron natural advantages which few if any places in the world surpass, the development of Canada's iron industry is wonderfully slow. There appears to be a good field for skill, enterprise, and capital in connection with our iron industry."

Sir Charles Tupper has recently said in the Canadian Parliament: "Our present annual consumption is equivalent to 250,000 tons of pig-iron, leaving steel rails out of the question altogether. To make this quantity of pig-iron it requires 750,000 tons of iron-ore, 120,000 tons of limestone, and 750,000 tons of coal to make it into iron in its first stage, pig-iron; and the freight required for the means of intercommunication in bringing these materials together amounts to not less than 1,625,000 tons. To manufacture it into puddled bars, merchant bars, and the various shapes and sizes into which it is made, would require an additional quantity of 750,000 tons of coal, making a total consumption of 1,500,000 tons."

On the ground that the development of the iron industry of Canada will tax the coal industry of the country to its utmost capacity, in order to furnish the additional output that will be required, and with all the advantages connected with that increased development, Sir Charles Tupper was enabled to say to the House that, although making anthracite coal free will take $497,000 away from the Government in revenue which it is now receiving, they would be perfectly justified in doing it, because the development of this iron-industry would be giving to the coal-mining industry a greater advantage and boon than that which would be taken away by the removal of the duty.

The disposition of the Government is further shown by the liberal bounty already mentioned, which is given, in addition to the protection afforded by the tariff, to producers of iron from Canadian ores. This bounty is payable quarterly in money.

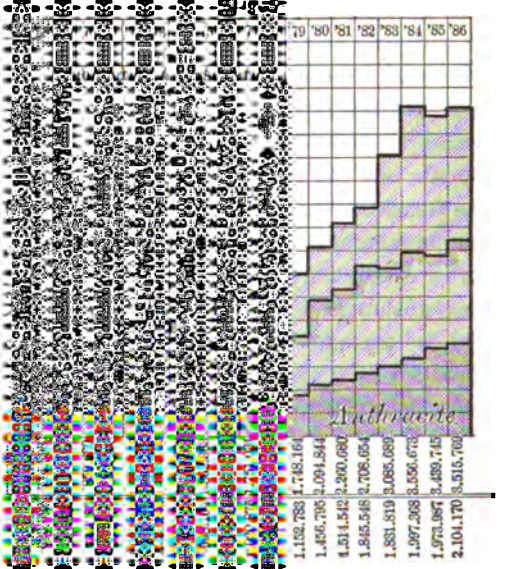
...ON TRADE.

...AL TRADE.

...adian coal-trade will be seen ...ns:

...*in the Dominion.*

	Net tons of 2000 lbs.
...	714,893
...	636,704
...	859,630
...	852,217
...	1,227,653
...	1,398,403
...	1,454,636
...	1,362,363
...	1,466,531
...	1,751,031
...	1,665,814
...	1,748,164
...	2,094,844
...	2,260,680
...	2,708,654
...	3,085,689
...	3,556,673
...	3,439,745
...	3,515,769



...represents the consumption; the next ...the consumption of imported anthra- ...of anthracite almost accounts for the

The total production of coal for the Dominion is shown in this table:

Total Production of Coal in the Dominion.

Year.	Net tons of 2000 lbs.
1868,	623,392
1869,	687,527
1870,	734,285
1871,	804,431
1872,	1,038,349
1873,	1,228,852
1874,	1,068,166
1875,	998,104
1876,	950,483
1877,	1,020,875
1878,	1,109,595
1879,	1,152,783
1880,	1,456,795
1881,	1,514,542
1882,	1,845,548
1883,	1,831,819
1884,	1,997,368
1885,	1,973,987
1886,	2,104,170

APPENDIX.

The following is the new Canadian tariff on iron and steel, and articles manufactured from them (the percentages stated are *ad valorem*, and the tons are net tons of 2000 pounds).

Goods Subject to Duties.—Wrought scrap-iron and scrap-steel, being waste or refuse wrought-iron or steel that has been in actual use and is fit only to be remanufactured, $2 per ton; ferro-manganese, ferro-silicon, spiegel, steel bloom-ends, and crop-ends of steel rails, for the manufacture of steel, $2 per ton; iron in pigs, iron kentledge and cast scrap iron, $4 per ton; iron in slabs, blooms, loops, puddled bars, or other forms less finished than iron in bars, and more advanced than pig-iron, except castings, $9 per ton; bar iron, rolled or hammered, comprising flats, rounds and squares, and bars and shapes of rolled iron, not elsewhere specified, $13 per ton; iron and steel wire, galvanized or not, No. 15 gauge and coarser, not elsewhere specified, 25 per cent. *ad valorem;* wire of spring-steel, coppered or tinned, No. 9 gauge or smaller, not elsewhere specified, 20 per cent.; boiler- or other plate-iron, sheared or unsheared, skelp-iron, sheared or rolled in grooves, and sheet-iron, common or black, not thinner than No. 20 gauge, not elsewhere specified, including nail plate of iron or steel, No. 16 gauge and thicker, $13 per ton; sheet-iron, common or black, smoothed or polished, and coated or galvanized, thinner than No. 20 gauge, Canada plates, and plates of iron or steel not less than 30 inches wide and not less than ¼ in. in thickness, 12½ per cent.; hoop or band or scroll or other iron, 8 in. or less in width, and not thinner than No. 20 gauge, $13 per ton; hoop or

of the Canadian militia; fire-brick, for use exclusively in processes of manufactures; anthracite coal; gannister; iron or steel rolled round wire-rods under ¼ in. in diameter, when imported by wire manufacturers for use in their factories; locomotive-tires of steel in the rough; rolled rods of steel under ½ in. in diameter or under ½ in. square, when imported by knob- or lock-manufacturers or cutlers for use exclusively in such manufactures in their own factories; steel rails, weighing not less than 25 pounds per linear yard, for use in railway tracks; steel valued at 2½ cents per pound and upwards for use in the manufacture of skates; scrap-iron and scrap-steel, old and fit only to be re-manufactured, being part of or recovered from any vessel wrecked in waters subject to the jurisdiction of Canada; steel-bowls for cream-separators; steel for the manufacture of files, when imported by file-manufacturers for use in their factories; wire of iron or steel, galvanized or tinned, No. 16 gauge or smaller.

The foregoing act went into force on the 13th May, 1887, and applied to all goods imported or taken out of warehouse for consumption on or after the said day; but all goods actually purchased on or before May 13th at any place out of Canada, for importation into Canada, on evidence to the satisfaction of the Minister of Customs of the purchase having been so made, and all goods in warehouse in Canada on that day, could be entered for duty until July 1st, or in the case of goods, at the rate of duty in force immediately previous. From the United Kingdom or any British possessions, carried by way of Cape Horn, to be entered in British Columbia until November 1st, 1887.

A NEW DISCOVERY OF CARBONATE IRON-ORE AT ENTERPRISE, MISS.

BY ALFRED F. BRAINERD, BIRMINGHAM, ALA.

(Duluth Meeting, July, 1887.)

A FEW months since, Prof. Lawrence C. Johnson, of the U. S. Survey, discovered in Mississippi large deposits of carbonate iron-ore, geologically located in the Claiborne formation of the Tertiary Epoch. These deposits are not pockets, but continuous layers or beds extending for miles, and can be located with certainty where they have not been removed by erosion.

Upon examination, recently made by G. W. West, Birmingham, Ala., the largest deposits and highest grades of ore were found near Enterprise, Lauderdale county, and in Clark county, Miss. These beds measure from 10 to 18 feet in thickness, are superimposed upon a "burr rock," associated with green sand, and dip 30 feet to the mile southeast, at an altitude of 350 feet above sea-level, can be

mined and economically worked by open-cutting after a few feet of stripping. In some places even this will not be found necessary, the ore being exposed at the surface. A number of analyses have been made by the writer and others, fully demonstrating the ore to be of sufficiently high grade to be marketable.

Ore Analysis.

	Iron.	Silica.	Phosphorus.
(1.)	31.82	40.30	0.382
(2.)	51.11	7.54	0.078
(3.)	42.82	19.62	.....
(4.)	39.62	20.52	.....
(5.)	14.75	.....	0.150
(6.)	46.80	.....	0.028
(7.)	30.12	36.00	0.620
(8.)	35.38	45.40	.....
(9.)	39.26	21.60	0.090
Average,	36.85	25.85	0.224

These analyses show the principal constituents of the ore except the carbonate of lime, which occur in small shells, rendering the material nearly self-fluxing.

No. 5 was selected as being high in lime, to ascertain whether it would be high in phosphorus and ought not to go in as ore.

No. 6 was made for D. McLaren, supt. M. & O. R. R. Analyst not known.

No. 7 was part of analysis made by Jos. Albricht, New Orleans, La., which also shows 5.50 carbonate of lime.

No. 8 was made by Chas. Mohr & Son, Mobile, Ala.

Roasting these ores dries all the volatile matter and increases the percentage of iron. The roasted ore was unfortunately, so far as I know, not analyzed.

Under the advice of the writer and Mr. West, about 60 tons of ore were roasted and shipped to Birmingham, and a furnace-run was made by Mr. B. T. Shelow, furnace-superintendent of the Alice furnace, yielding an output of 40 per cent. of No. 3 foundry-iron. In this test, the mixture was one-half Red Mountain soft ore, and one-half Enterprise carbonate ore. The iron compares favorably with No. 3 foundry-iron made two years ago at the same place, but with different cinder and burden.

The analyses are as follows:

Iron Analysis.

	I.	II.
Total carbon,	2.488	4.076
Silicon,	3.549	3.870
Phosphorus,	0.717	0.489
Sulphur,	0.242	0.040
Manganese,	0.143	0.389
Iron (by difference),	92.861	91.136
	100.000	100.000

I. No. 3 foundry-iron made at Alice furnace with ½ Enterprise carbonate ore, 1887.

II. No. 3 foundry-iron, made at Alice furnace, with ½ soft and ½ hard, red ore, 1885.

The following cinder-analyses are also given for comparison :

Slag Analysis.

	I.	II.	III.
Protoxide of iron,	0.270	0.436	0.60*
Silica,	45.460	35.000	33.00
Alumina,	16.690	14.362	14.00
Lime,	32.805	45.370	49.00
Magnesia,	1.080	1.398	1.00
Binoxide of manganese,	0.083	trace	.....
Sulphur,) Sulphide of	1.571	1.875	2.00
Calcium,) calcium,	1.963	1.500	.....
Phosphoric acid,	0.008	0.059	.....
Undetermined loss,	0.070	.....	.....
	100.000	100.000	

I. Slag made during the run of Alice furnace on mixture containing Enterprise ore.

II. Slag made at the Sloss furnace in June, 1886, on No. 1 foundry iron.

III. Average constitution of slag made at Alice furnace in 1885, when producing iron such as is shown in iron analysis II. above.

Such a slag as II., though frequently made for months at a time by Birmingham furnaces, can scarcely be called typical of their practice. The endeavor now is to get higher silica, whereas formerly 31 to 32 per cent. silica and 13 to 15 alumina was rather the rule than the exception.

The figures under III. are estimated from many analyses, and probably represent fairly the cinder accompanying pig-iron II above. Unfortunately no analysis was made of that particular cinder. It will be seen that slag I is much more siliceous than III; in fact I

* Oxide of iron and manganese.

never before heard of a cinder produced in this district with so much as 45 per cent. of silica. As the iron in the two cases was of similar grade and quality, it seems clear from these analyses that highly siliceous slags, involving less lime, and therefore a smaller burden, could be used in the Birmingham district with a gain in fuel-economy. The ordinary Alice furnace-slag now ranges high in silica.

With regard to the deposits at Enterprise, it may be added that they are advantageously situated for development. An abundant supply of water may be obtained from several creeks; there are in the immediate neighborhood immense primitive pine forests, suitable for both charcoal and lumber; and, finally, two trunk lines of railroad, the Queen and Crescent and the Mobile and Ohio, run through Enterprise, contiguous to the ore-beds, which lie between them and are easily accessible to both.

EXPERIMENTS ILLUSTRATING THE DESCENT OF THE CHARGE IN AN IRON BLAST-FURNACE.

BY ROBERT H. RICHARDS AND RICHARD W. LODGE, OF THE MASS. INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Duluth Meeting, July, 1887.)

A GREAT DEAL of speculation, as well as actual experiment, has been devoted to ascertaining the changes in the materials during their descent in an iron blast-furnace, affecting (1) the chemical constitution, (2) the mechanical condition, and (3) the arrangement of the materials. With a view to throw some light upon the third question, we constructed a model and tried experiments which are here described.

Our model is a vertical section of a furnace, with the lines of the Edgar Thomson furnace D of 1885, for the use of which we are indebted to Mr. Gayley. The model is 40 inches high, and made to scale ($\frac{1}{2}$ inch equals 1 foot), and the horizontal section of the space through which the charge falls at any point in the descent is $1\frac{1}{2}$ in. $\times$ d, d being the diameter of the furnace at that place.

We are unable to say just where the idea of making this model came from; perhaps, partly from *The Principles of the Manufacture of Iron and Steel*, 1884, I. Lowthian Bell, p. 124; partly, also, from

an indistinct recollection of some one's having reported to us that he had made such a model. The facts, however, are gone from memory now.

The back and sides of the model are of $\frac{7}{8}$-inch pine board; the front is $\frac{1}{4}$-inch plate-glass, screwed to the sides. The space between the glass front and the board back is $1\frac{1}{2}$ inch. Below is a little bin into which the charge falls, and from which it is shoveled back to the bell-and-hopper. Every shovelful withdrawn from below settles the contents of the stack sufficiently to receive a new charge above, and in this way a great variety of feeds may be speedily tried, and the arrangement due to them inspected through the plate-glass.

The material used for the charge in these tests was composed of beach pebbles and clean quartz-mill sand; and the different sizes were mixed in the proportion indicated below, making four lots:

	Lot I. Per cent.	Lot II. Per cent.	Lot III. Per cent.	Lot IV. Per cent.
Through $\frac{1}{2}$-inch sieve on $\frac{1}{4}$-inch sieve,	38.74	51.66	64.0	76.08
Through $\frac{1}{4}$-inch sieve on $\frac{1}{8}$-inch sieve,	2.56	1.81	2.0	2.43
Through $\frac{1}{8}$-inch sieve on $\frac{1}{16}$-inch sieve,	13.67	11.17	9.0	6.39
Through $\frac{1}{16}$-inch sieve on $\frac{1}{30}$-inch sieve,	4.56	3.32	2.3	1.40
Through $\frac{1}{30}$-inch sieve,	40.46	32.02	22.70	13.69

Each lot of material, so mixed, was used for a series of tests, and, in every case, as soon as the furnace had got into normal running, a photograph of the condition of the charge at that time was taken. From these photographs thus obtained the accompanying admirable engravings were made by the wax-process. They have been carefully compared with the originals and found to be accurate. These illustrations are in series, with two variables, namely, first, the kind of the bell or distributor, and secondly, the proportion of coarse and fine in the material used.

The Roman numerals, I, II, III, IV, are used to designate the lot of material referred to, and the letters, *a*, *b*, *c*, etc., are used to designate the bells or distributors which were used—*a*, large bell; *b*, small bell; *c*, no bell at all.

Figs. I, II, III, and IV, with variable gravel in series, will be seen to start with a large proportion of fine in I, and gradually become coarser till IV is reached, in which there is a large proportion of coarse.

Figs. *a*, *b*, *c*, with variable bell in series, will be seen to start with *a* (large bell), having three columns placed thus—a column of coarse

material in the center with fine at the two sides; then comes *b* (small bell), with five columns, having coarse in the center and an annulus of fine, representing two intermediate columns and two columns of coarse again on the outside; finally, *c* (no bell), which is fairly well mixed, except that there is a trifling excess of coarse material on the margin.

Fig. I*f* has been taken since the paper was written; it represents a furnace with a funnel delivering the whole of the gravel at the center of the stock-line, and in it will be found two columns of coarse on the sides and a column of fine in the center.

Figs. *d* and *e* were only tried for one series, and they go to show that with a small bell, if the stock is not kept up to the proper stock-line, the arrangement approaches *a*; in *e*, nearly so, and in *d*, quite so; *i.e.*, a small bell poorly fed does the work of a large bell well fed, at the expense, however, of loss of height in the furnace. See II*a*, II*d*, II*e*.

The large bell is the normal Edgar Thomson bell; the small bell is 4 inches (representing 8 feet) in diameter, and was chosen not from any known practical proportions, but simply to see what would be the effect of a small bell.

Series I shows isochronous lines, which connect particles that have been in the furnace the same length of time. These curves were obtained by putting in a layer of charcoal and chalking its position after every withdrawal of charge from below.

In examining these photographs, we notice:

1. That a bell which is large, with reference to its stock-line, always gives the charge in three columns, the two outer are fine, the one inner column is coarse. See I*a*, II*a*, III*a*, IV*a*.

2. That a bell which is small, with reference to the stock-line, and which is kept full fed, always yields the charge in five columns. The center is coarse, then an annulus of fine, and then margins of coarse again. See I*b*, II*b*, III*b*, IV*b*.

3. That a furnace fed without a bell, and whose top is smaller than the stock-line, will give a fairly even mixture; the coarse material being slightly concentrated in the margin. See I*c*, II*c*, III*c*, IV*c*. And that a furnace fed with a central funnel gives three columns, namely, fine in the center and two coarse columns on the sides. See I*f*.

4. That if the stock-line be varied from high to low, with a small bell, the arrangement of particles resulting will vary from the five columns to three columns. See II*b*, II*e*, II*d*.

5. Columns of coarse material appear to settle more rapidly; columns of fine, less rapidly. See I *a*, I *b*, I *c*, I *f*. Here the coarse in the center of I *a* appears to descend fully twice as fast as the fine on the sides. In I *b* the coarse being on the sides as well as in the center, the charge settles about evenly until the point is reached where the friction and squeezing of the boshes comes in, when the central coarse column pushes ahead. In I *c*, the coarse is slightly concentrated in the margin, and hence the margin settles a little faster until the boshes are reached; and finally, in I *f* the sides are wholly coarse while the center is fine, and here we have the sides gaining on the center a little faster even than in I *c*.

If we are rightly informed, there is a general opinion among ironmen that a hand-fed furnace is better fed than any other, but that the quantities now used forbid hand feed. The evenness of the feed in I *c*, II *c*, III *c* and IV *c*, would seem to bear out this opinion. Again, in order to drive a furnace for great production, it is necessary to have a large bell (that is to say large in the sense used in this paper), and one with a proper proportion to the stock-line. Now the greatest danger in a hard-driven furnace would presumably be that the lining would give out. Observe how the work is balanced for fast driving and brick-saving in I *a*, II *a*, III *a*, IV *a*. Here the side columns are fine, the gases penetrate slowly, reduction is slow, combustion is slow, heat cannot attain its highest degree and the bricks are saved. To balance all this, and prevent the furnace from going to white iron and black slag, this part of the charge moves slowly down through the furnace and thus has longer time to pass through its chemical and physical changes. On the other hand, the center of the charge moves rapidly down, has rapidly moving gases, combustion is rapid, reduction is rapid, heat attains a high degree, and hence the ore can safely pass through its chemical and physical changes, even though the time allowed it is much shorter. If this idea be correct, then hand-feed can only be applied to slow-running furnaces, for the coarse will be at the sides (I *c*, II *c*, III *c*, IV *c*), and, with hard driving and great production, will fail to protect the brick linings.

We presume that the forming of coarse and fine columns will have a tendency to produce an "unmixing" of the charge. While at times this might cause trouble, it again might give the furnace-man a chance to put in his medicine where it was wanted. For instance, he could put more lime in the center and less on the sides, or *vice versa*, by having the limestone coarser in the first instance and finer in the

second. The same would be true also of the coke. It will take a person of more experience in iron than either of us to tell whether scaffolds could be removed by feeding, for several successive charges, three-quarters of the bell with normal ore-charge, and the other one quarter, immediately over the scaffold, with all-coarse charge, and so for several successive charges to lead the high heat over to the side where the scaffold was located, and in that way to partly scour and partly burn the obstruction off.

Fig. I *g* was obtained while the model was being designed. The discharge at the bottom was only 1½ inches wide at that time, and the figure is put in simply to show how a perfectly stationary scaffold can form in this model bed with dry gravel and sand, if only a little obstruction is left for it to rest upon. All the space behind the chalk line is scaffold.

We presume that the remark will occur to the reader that "a forty-inch furnace cannot reproduce exactly the conditions of an eighty-foot furnace." The following exceptions may be taken to experiments of this character:

1. A rectangular horizontal section gives an untrue representation of the two opposite sectors taken from a circular horizontal section. The error here is that the central columns shown in our diagrams are a little too narrow and the side columns are a little too wide.

2. Our material is all of the same specific gravity, while iron-ore, limestone, and coke vary a good deal. This might make a difference in the arrangement, but probably not much, as the fragments take their places according to size, the largest rolling the furthest, the smallest lodging almost immediately.

3. ½-inch and ¼-inch pebbles may be mathematically right for a 40-inch furnace as compared with the usual size, fed to an 80-foot furnace, but cannot of course exactly reproduce the effects.

4. The chute of the material with an 11-foot bell may differ in velocity from that of a 5-inch bell, but in each case they simply deliver the material at the angle due to the bell which is the same in both cases.

5. In a rectangular sectional model the friction of the charge on glass in front and wood behind, would cause a different condition from that in a circular furnace, tending to retard the rapid-moving particles more than the slow, and in this way flattening the isochronous curves.

6. The loosening of the charge in the large furnace by the combustion of the coke cannot be represented in the model. If the coke

IRON BLAST-FURNACE.

FIG. I *b*.

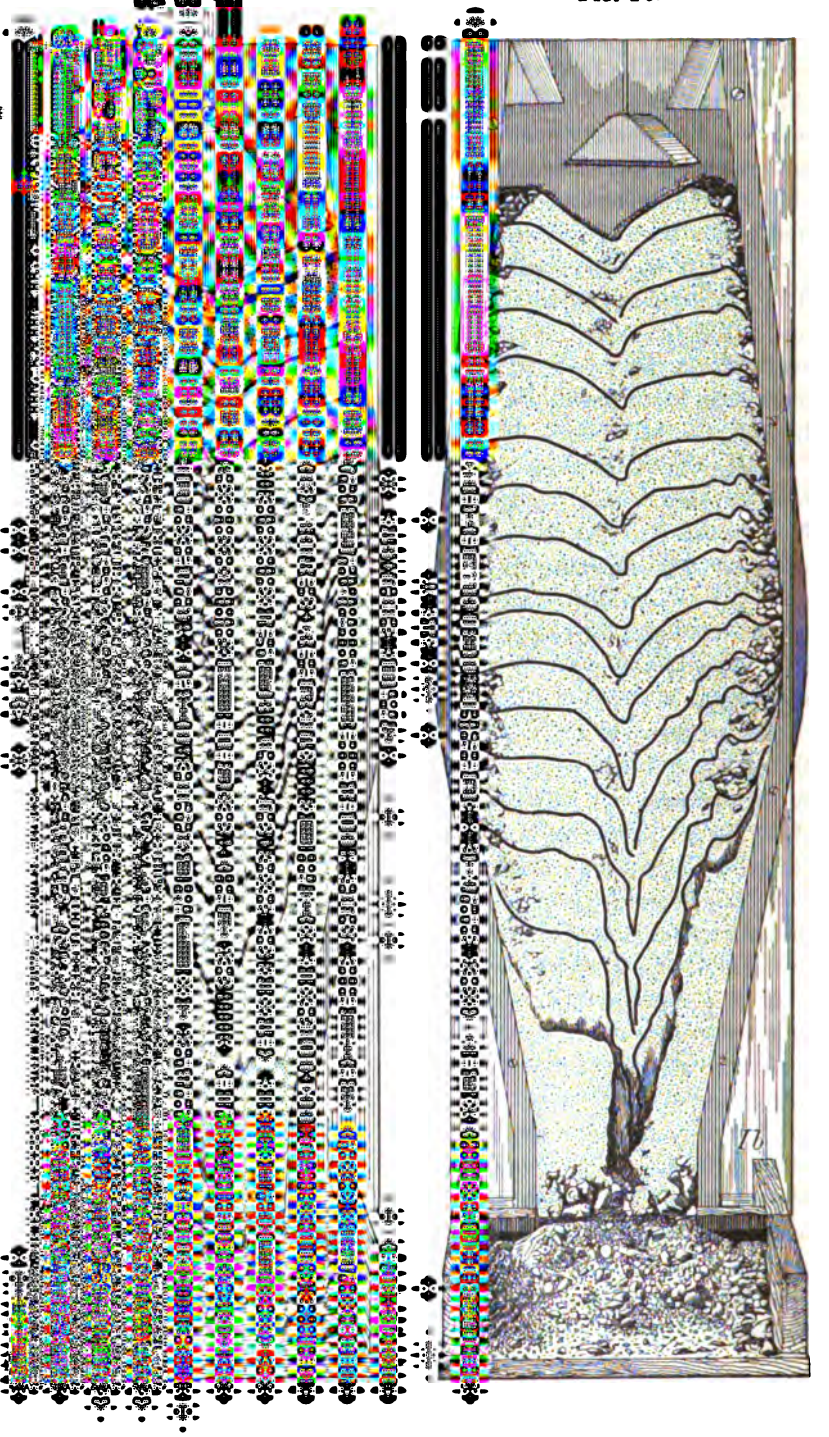
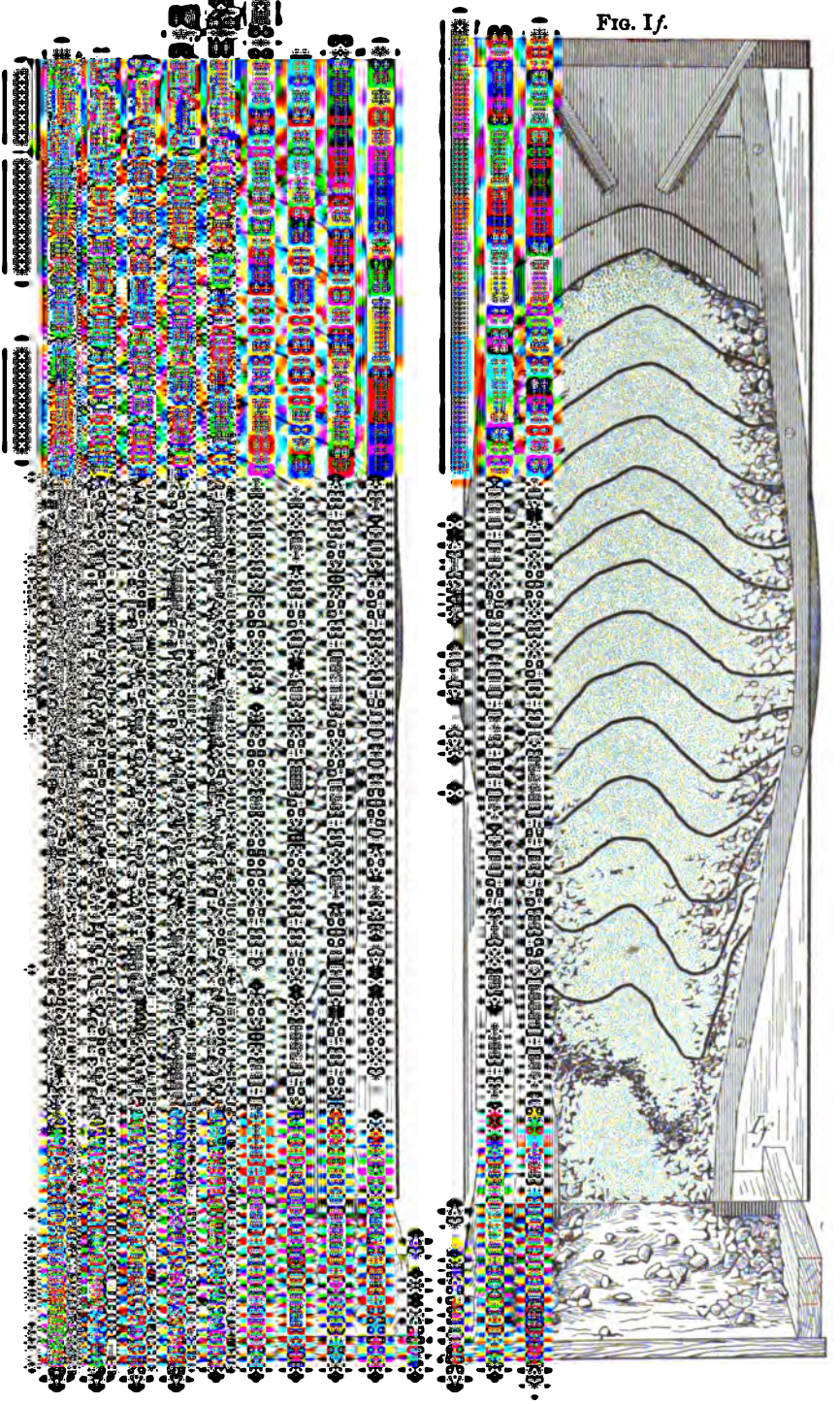
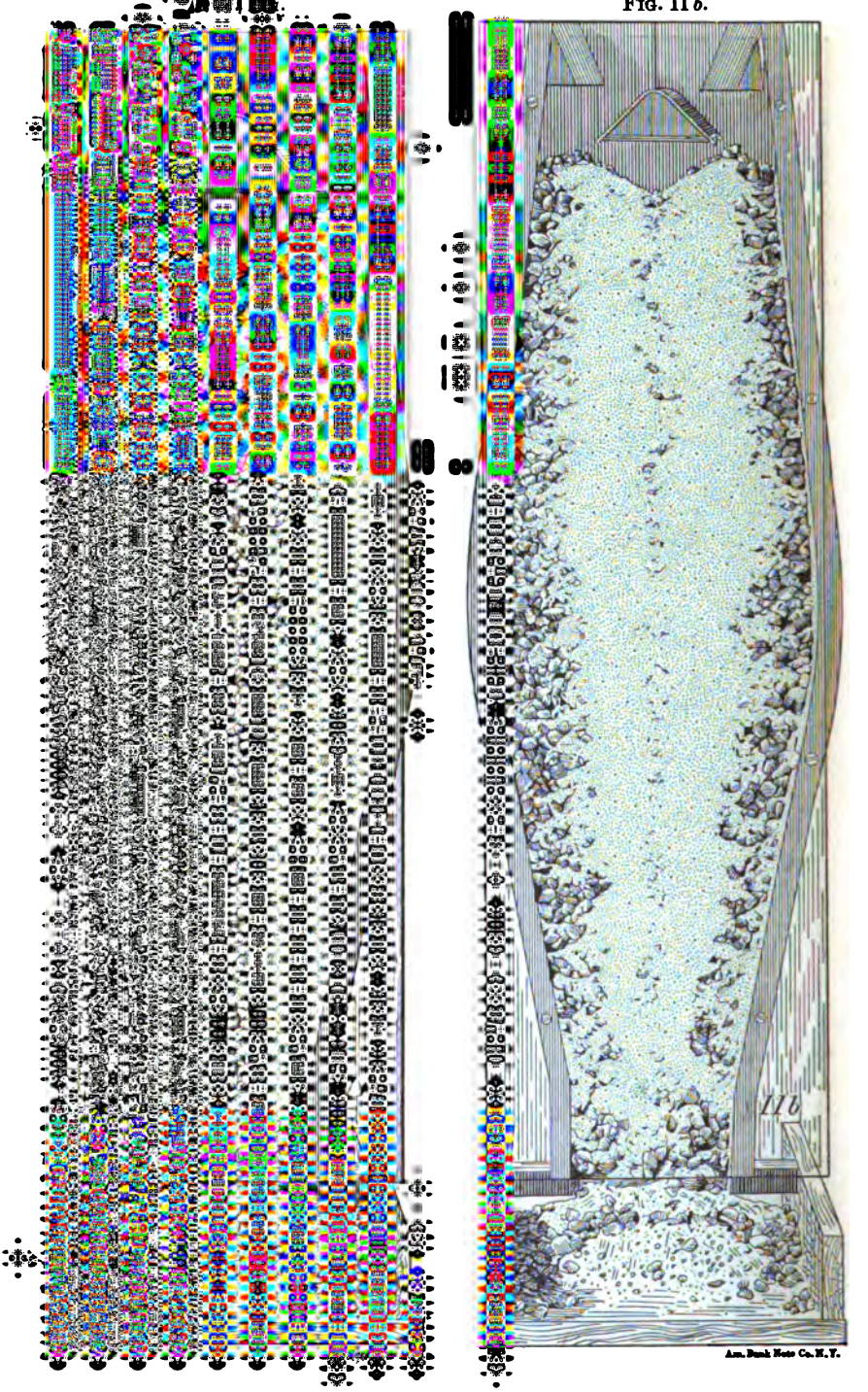


FIG. I*f*.



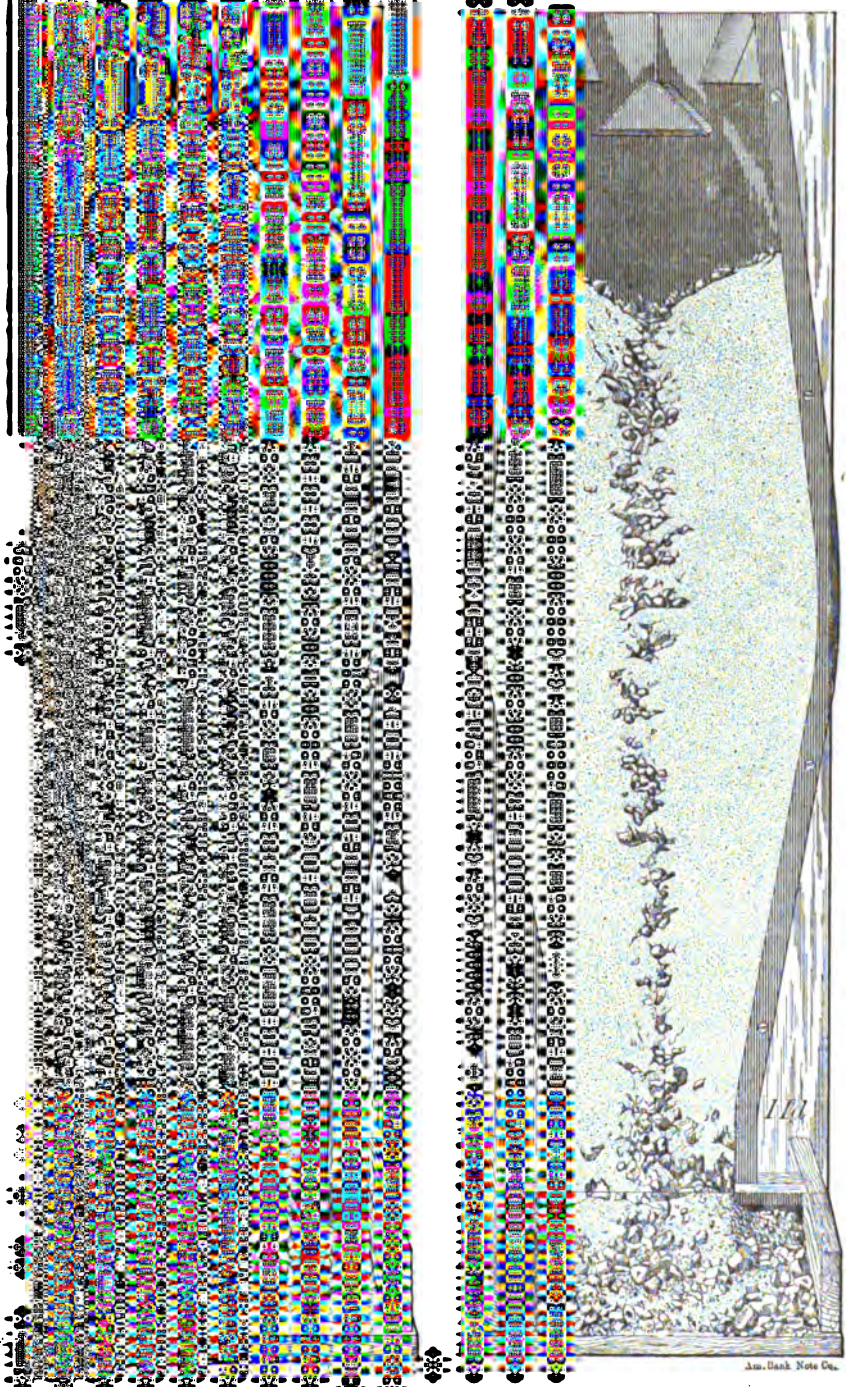
IRON BLAST-FURNACE.

FIG. II *b*.



Am. Bank Note Co. N. Y.

FIG. II *d*.



N IRON BLAST-FURNACE.

FIG. III *b*.

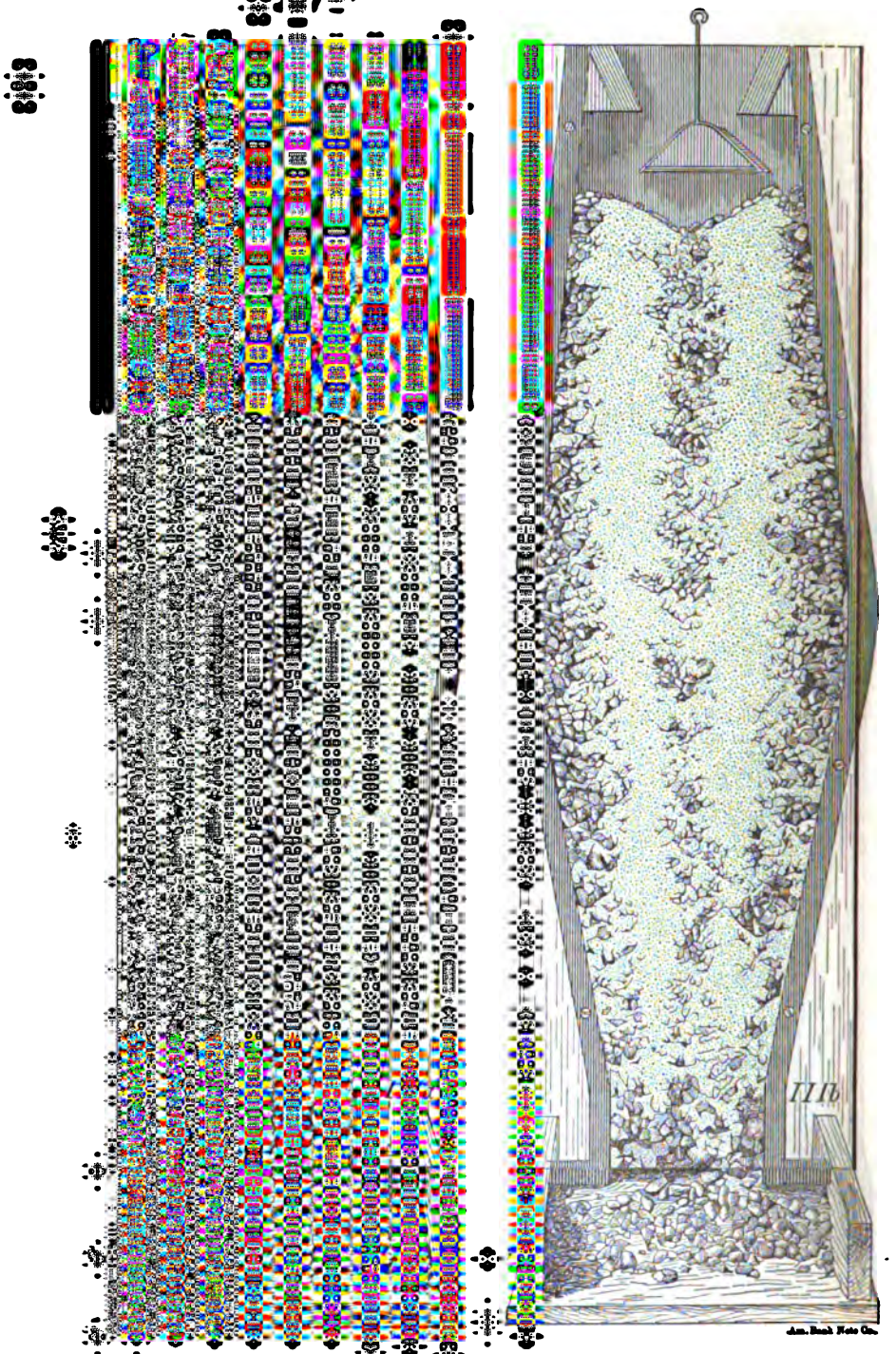


FIG. II e.

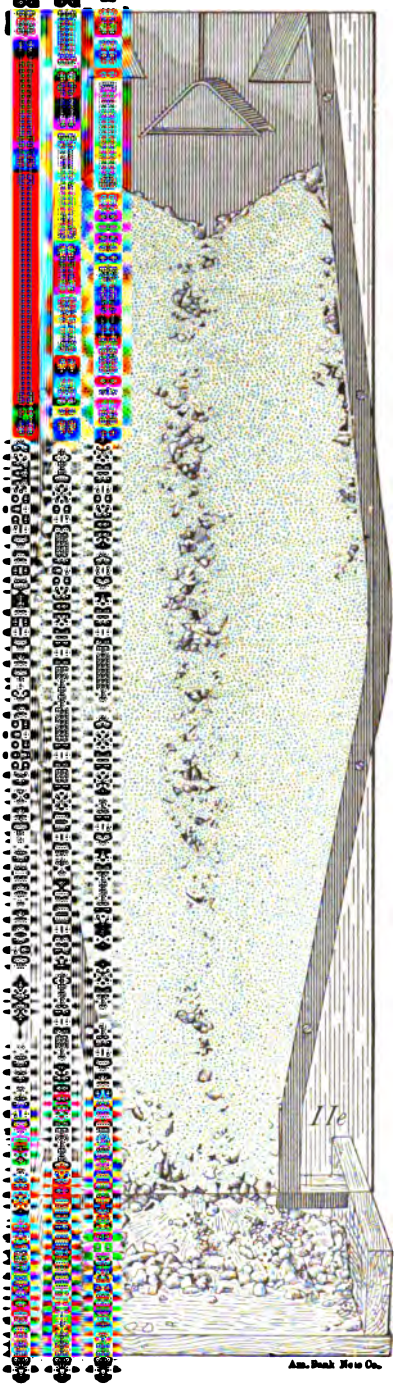


Fig. IV *b*.

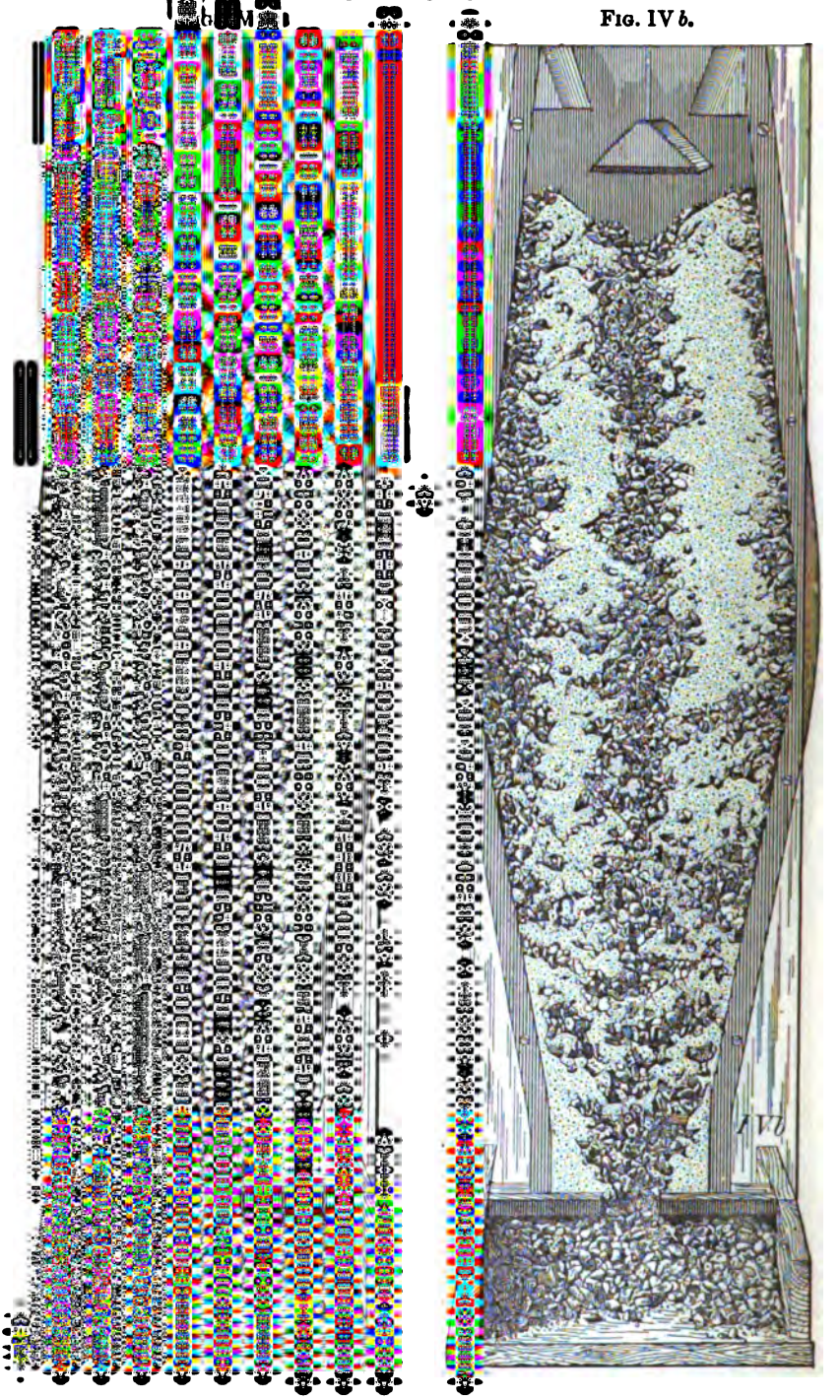
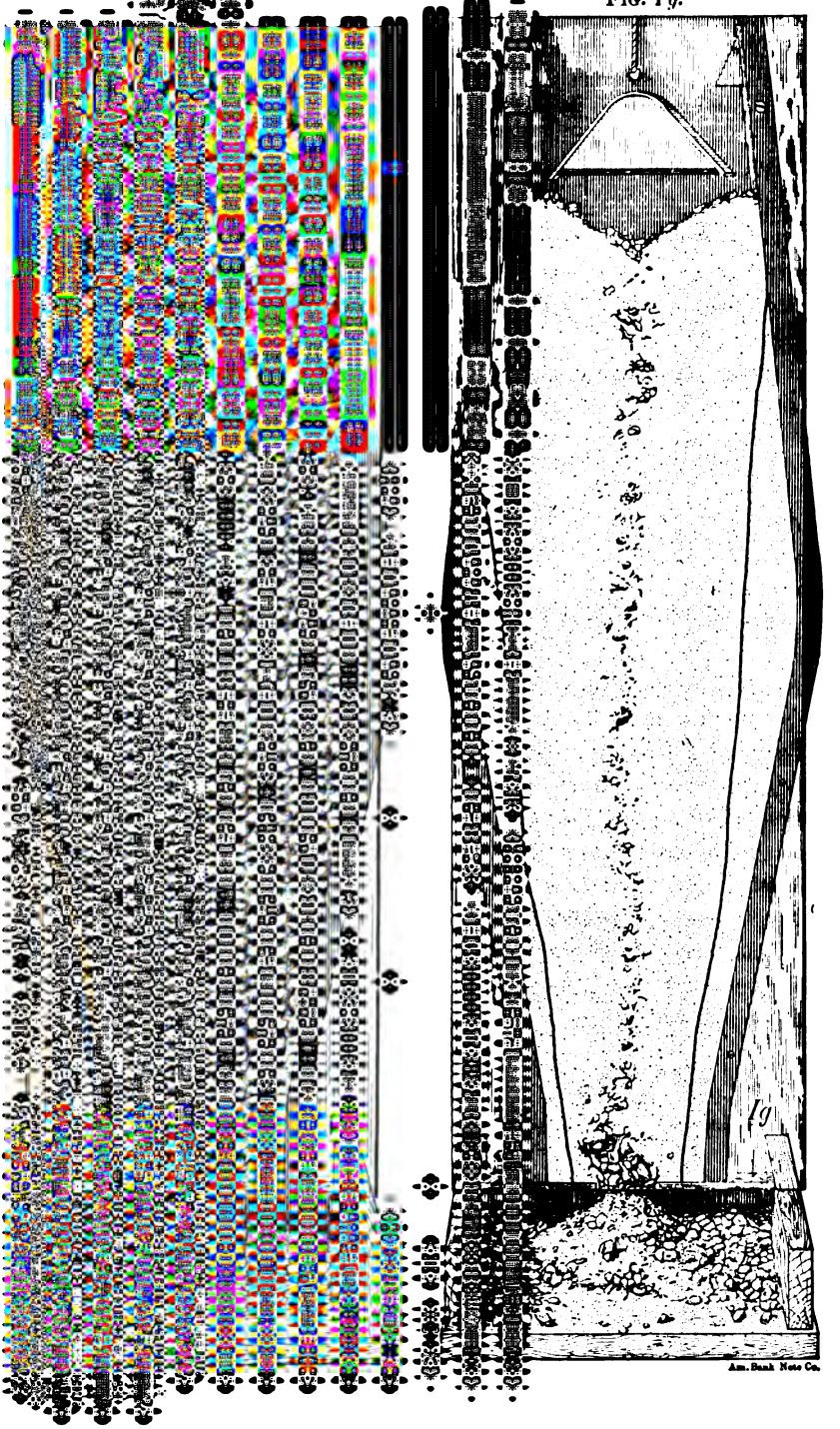


FIG. I g.



were evenly mixed this would affect the settling evenly, but if, as is more likely the case, the coke is somewhat concentrated in the center then the center would be accelerated. This fact would cause the isochronous curves in I *c*, I *f*, to be too convex and those in I *a*, I *b*, to be too obtuse in the model.

We presume that exception No. 1 makes a noticeable difference, narrowing the inner columns and widening the outer; and that Nos. 5 and 6 both cause a greater retarding of the fast-moving, as compared with the slow-moving particles, in this way giving flatter isochronous curves than would be true in the large furnace; nevertheless, crude and defective as the experiments are, we hope they may throw some light upon the difficult question of furnace-feeding.

GILSONITE OR UINTAHITE, A NEW VARIETY OF ASPHALTUM FROM THE UINTAH MOUNTAINS, UTAH.

BY JOSEPH M. LOCKE, C.E., M.E., SALT LAKE CITY, UTAH.

(Utah and Montana Meeting, July, 1887.)

THE discovery of this asphaltum was made by S. H. Gilson, of Salt Lake, and since then the material has borne the local name of Gilsonite. So far as I have been able to ascertain, however, the first *published* notice of this discovery was made by Prof. W. P. Blake in the *Engineering and Mining Journal*, of December 26, 1885, where he names the new mineral Uintahite, from the place of its discovery, the Uintah Mountains in eastern Utah. The priority of publication and the geographical fitness of the name will probably secure its acceptance by mineralogists.* The following is Prof. Blake's article.

In the month of September last, a sample of peculiar asphalt was received for examination, and recently when in Provo, Utah, I had the opportunity of seeing a considerable quantity brought in from the Uintah Mountains.

It is obtained in masses several inches in diameter, and apparently free from mechanically disseminated impurities. It breaks with a conchoidal fracture, is very brittle, and is readily reduced to powder in a mortar. Hardness, 2 to 25. Gravity, 1.065 to 1.070. Color, black, brilliant, and lustrous; streak and powder, a rich brown. It is a non-conductor of electricity, and is electrically excited by friction.

* NOTE BY THE SECRETARY.— *Uintite* would be a more agreeable form. In this paper the name Gilsonite, originally used by Mr. Locke, has been retained.

It fuses easily in the flame of a candle and burns with a brilliant flame, much like sealing-wax; and, like sealing-wax, it will give a clean, sharp impression from a seal. Unless the melted mineral is very hot, it does not adhere to cold paper. It has considerable plasticity while warm, and is not sticky, but retains after melting its lustrous black and smooth surface. By distillation, a very small quantity of a clear, white and dense oil is given off, and a little gas or vapor.

It is much more readily dissolved by the heavy oils and fats than by the lighter and more volatile menstrua. Thus it dissolves and incorporates quickly in heavy lubricating petroleum, while the white distillates from petroleum have little or no effect upon it at ordinary temperature. So also it freely dissolves in oil of turpentine, when warmed, but it does not readily dissolve in cold spirits of turpentine. Ether apparently does not attack fragments, but the powder is slowly dissolved. It is not soluble in ordinary alcohol.

The mineral dissolves quickly in melted wax and incorporates with it, producing on cooling a hard, black mixture, which closely resembles "burnt wax," and may be used for similar purposes. The mixture also closely resembles the Utah mineral wax or ozokerite. The mineral will also freely melt and combine with ozokerite. It dissolves in hot stearine and melted tallow, giving black mixtures varying in hardness with the amount of uintahite used.

It differs from Grahamite and Albertite by its easy fusibility and other characters; from piauzite by its gravity, structure and deportment with alcohol; and from wolongonzite by its fusibility and small amount of oil and gas.

It is said to occur in considerable quantities, and it will no doubt find many useful applications in the arts as a pigment, insulator and ingredient in lubricating cereous and fatty mixtures.

The analysis of uintahite was made by Messrs. E. T. Fristol and W. P. Lawver, chemists and assayers, at the Columbian University, Washington, D. C.

Carbon,	78.43
Hydrogen,	10.20
Nitrogen,	2.27
Oxygen,	8.70
Ash,	.40
	100.00

Soluble as follows: Bisulphide of carbon and chloroform dissolve it completely; benzole dissolves 95 per cent.; ether, 86.5 per cent.; absolute alcohol, 9.5 per cent.

The following notes upon some of the properties and uses of Gilsonite are in no sense the result of an exhaustive and systematic examination, but embrace conclusions from a large number of experiments, chiefly of a technological character.

Gilsonite is a lustrous, black, friable solid with conchoidal fracture, inodorous, and when pulverized, yielding a dark brown powder. It has a specific gravity at 62° F. of 1.06051 and 1.05742 and 1, or an average of 1.05896, and consequently weighs 62.655 pounds per cubic foot. It does not absorb moisture. It is a true hydrocarbon; the ash (*i.e.*, earthy matter) obtained by combustion is not a constant quantity in various samples, being simply due to the presence of more or less foreign matter. It is a non-conductor

of heat and electricity to a remarkable degree. Naphtha and sulphuric ether soften it with intumescence; alcohol, mineral acids and solutions of alkalies, at ordinary temperatures, have no effect upon it; chloroform and bisulphide of carbon dissolve it at ordinary temperatures, the latter not always completely. In common with ordinary resin and other materials of its class, it appears to be composed of two ingredients, one of which, in some solvents, is more readily dissolved than the other; but the line of demarcation of solubility is so indefinite as not to permit ready separation of the two component parts, nor is there any technical reason, so far as I know, for desiring such a separation.

Vegetable, animal and the heavier petroleum oils at ordinary temperatures soften Gilsonite and, under slight elevation of temperature, unite with it in any proportion.

Spirit of turpentine at ordinary temperatures readily dissolves one of the component parts, the other remaining as a slightly gelatinous mass, largely impregnated with the solution of the first portion, but a slight application of heat results in complete solution, which remains permanent upon cooling. Spirit of turpentine and Gilsonite readily unite in any proportion.

Heat softens Gilsonite without melting it; but upon raising the temperature a little over 500° F., a portion of the Gilsonite undergoes destructive distillation (similar to that of coal in gas-retorts), during which operation there are evolved gas and hydrocarbon oils, the latter of which, being solvents of Gilsonite, soften the remaining unchanged portion of that substance. The mixture thus formed is fusible, but upon cooling is no longer hard and brittle, but more or less soft and tough, as there is more or less of the oils in the mixture.

The term asphaltum, as now generally applied, designates petroleum that has been more or less indurated by exposure to the air, and is composed of two portions: 1st, asphaltene, a black, brittle mass which does not melt, but softens and decomposes under heat; 2d, petrolene, an oily portion which imparts to the original material more or less softness, according to the greater or less proportion of petrolene. The petrolene can be distilled off from the asphaltene by heat. But the name asphaltum was originally applied to an article found floating upon the surface of the Dead Sea (anciently called Asphaltites), from which source most of the supply was obtained. This substance is apparently identical with Gilsonite. The origin of the asphaltum of the Dead Sea had generally, but un-

satisfactorily, been attributed to petroleum, becoming indurated as it exuded from the rocks at the bottom of the sea. A more rational explanation is furnished by the finding of Gilsonite in a true fissure-vein, which permits us to attribute to the Dead Sea article a similar origin.

In order to compound Gilsonite with oils, coal tar, paraffine or similar materials, by the aid of heat, it is necessary to pulverize the Gilsonite, and then mix intimately with the other ingredients before heating, and upon application of the heat, care must be taken not to raise the temperature above about 500° F., otherwise the Gilsonite will be partially decomposed, thus destroying more or less the characteristic of the compounds. In practice this is the chief cause of a want of uniformity in such preparations; therefore this danger of over-heating cannot be too carefully guarded against. A series of experiments was made to determine what amount of oil is necessary, in order to give the Gilsonite sufficient tenacity, and extend the toughening effect to a sufficiently low temperature, without, on the other hand, unduly lowering the temperature at which the compound would become too soft.

The results obtained from the two samples which approximated most nearly to the desired conditions, are shown in the following:

TABLE I.

Composition.		Consistency under various temperatures. (Fahrenheit Scale.)							Remarks.
Gilsonite. Per cent.	Linseed Oil. Per cent.	Melts.	Commences to stiffen.	Is still soft.	Can be impressed with finger-nail, but is not sticky.	Is waxy.	Is readily cut by the finger-nail.	Is hard.	
83⅓	16	365°	325°	230°	20 ,°	190°	180°	100°	At 85° rather hard and brittle.
66⅔	33⅓	295°	265°	212°	160	140°	130°	100°	At 85° can be slightly indented by finger-na

Spirit of turpentine appears to be the commercial solvent for Gilsonite, and when mixed in the proportion of one pound of Gilsonite to five pints of turpentine, and gently heated, the result is an excellent japanning varnish, which, upon being applied to metallic surfaces and then baked, becomes quite hard. The same varnish with the addition of ½ pint of oil can be applied to fabrics to

render them waterproof. The article will remain flexible after the varnish has become fully dried. Unless several coats are applied, so as to completely obliterate all appearance of the texture of the fabric, the fibre will not be sufficiently protected to resist the action of acids to which it may be exposed. Articles so prepared cannot in all cases be substituted for India-rubber goods, as the material, whilst sufficiently flexible, is too soft to resist wear; nor can it be toughened by vulcanization, as the only article acted upon by such a process is the oil (linseed) in the compound, and this action is slight. No vulcanizing effect is produced upon the Gilsonite.

A series of sand-mastics were prepared by mixing and pulverizing Gilsonite intimately, then adding oil and continuing the mixing until the materials were thoroughly incorporated, having a consistence from rather dry to even wet moulding-sand. Each batch of the mixed materials was then placed in a vessel and gradually heated, not allowing the heat to become sufficient (*i.e.*, over 500° F.) to decompose the particles of Gilsonite in contact with the vessel. When the mass was heated throughout, test-bars were formed by placing it in a warm mould, where it remained under a pressure of about 6000 pounds per square inch, until it was cold. From the test-bars several were selected for further investigation, the results of which are shown in Table II.

TABLE II.

Designation mark of sample.	Composition in percentage of each ingredient.					Breaking strain in pounds of test-bar 1 inch square, and 4 inches between bearings, with force applied at center. Temperature of bar at time of experiment. — Fahrenheit degrees.			Crushing strain in pounds of test-piece, 1 inch cube. Temperature of test-piece at time of experiment. — Fahrenheit degrees.			Amount of water absorbed to saturation in percentage of weight of compound.
	Whitish sand from near Salt Lake glass-works.	Jordan River sand from near Bingham Canon R. R. bridge.	Pit-sand, near Old Telegraph mill at West Jordan.	Gilsonite.	Linseed Oil.	120°	78°	18°	120°	78°	18°	
A	88.24			9.8	1.96		50		437	911		1.92
B	80			15.4	4.60		114		1989	1707	2187	0.39
C		85		11 55	3.45		32		465	456		2.90
D		80		15.40	4.60		92		442	449		0.50
E		75		19.25	5.75	67	87	165	290	494	667	0.58
F		58		32.30	9.70		63		207	416		0.41
G			80	15.4	4.60	78	74	133	445	582	1095	0.21
H			75	19.25	5.75		64		235	407		0.13
I			64	27.72	8.28		42		277	435		0.18

Before discussing these results, attention should be drawn to the fact that only one experiment was tried under each condition for breaking and crushing strain in lieu of a number, the average of which could then be taken. Nevertheless, these figures are sufficiently accurate to give a general idea of the relative merits of the bars, and their actual strength.

The sand from near the glass-works is clean and sharp, and contains over 95 per cent. of silica.

The sand from Jordan river is clean, not very sharp, and carries a considerable amount of carbonate of lime.

The pit-sand is neither sharp nor clean, and carries the largest amount of carbonate of lime.

The last column, in which is given the percentage of water absorbed by saturation, indicates the amount of liability to disintegration by frost and atmospheric agencies.

We may deduce from Table II. the following conclusions:

1st. The strongest mastic is obtained by the use of about 80 per cent. of sand.

2d. The character of the sand exerts considerable influence upon the strength of the mastic, the strength being greater as the sand is sharp and clean.

3d. Decreasing the sand below 80 per cent. renders the mastic more susceptible to change of breaking and crushing strain by variations of temperature.

4th. The power to absorb water decreases as the percentum of Gilsonite is increased.

Among the practical uses of this substance we may mention the following:

Composition roofing.—As the principal ingredient in such compounds it possesses superior qualities.

Asphalt pavements.—For this purpose it is a very superior material, and will fully meet the exacting requirements of the present day, but must be prepared as before stated when speaking of mastic. To determine the proportion of mastic to broken stone or gravel which is required, pack the latter firmly in a vessel, and add water until the interstices are all well filled. The amount of water required to accomplish this will equal the amount of mastic necessary for the quantity of rock or gravel experimented upon. The broken stone or gravel can be mixed with the ingredients of the mastic before heating, or beaten into the mastic in place whilst still hot, in which case the coarse material must be hot enough to melt the mastic, but not decompose it.

Electrical purposes.—The electrical conductivity of Gilsonite being exceedingly small, it is a valuable insulator; and its non-absorption of water adds greatly to its value in the construction of underground conduits.

In preparing a compound for insulation, I would use 70 per cent. sand and 30 per cent. mastic, the latter being composed of 83⅓ per cent. Gilsonite, and 16 per cent. oil.

Compounding with ebonite.—For this class of manufacture, Gilsonite is equal to any known asphaltum.

THE RESOURCES OF THE LAKE SUPERIOR REGION.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

(Duluth Meeting, July, 1887.)

In the belief that a *résumé* of what will be exhibited during this meeting, and a brief record of progress in the seven years which have elapsed since the Institute's first visit to Lake Superior would be interesting, the following statistical data have been prepared or copied from the most reliable sources, and considerations are also presented concerning the development or utilization of some of the resources of the Lake Superior Region.

When the Institute, on the return trip from enjoying the charming panoramas of Thunder Cape or Isle Royale, and inspecting the unique development of Silver Islet, passed, in August, 1880, through the St. Mary's Falls ship canal, en route to Escanaba, the boat was lowered from the level of Lake Superior to that of Michigan and Huron lakes (18 feet), through two locks each 70 feet wide and 350 feet long, which had been in use since 1855. But as these locks accommodated vessels which drew 12 feet of water or less, a new lock 80 feet wide, 515 feet long and admitting vessels of 16 feet draught was approaching completion and was opened in September, 1881. The time required to pass the locks was utilized by the members in examining the work upon this new lock and in viewing the Sault Ste. Marie, which in discharging from Lake Superior an average volume of 90,783 cubic feet of water per second, forms a rapid, falling 18 feet in one mile. But the lake traffic through the St. Mary's ship-canal has developed so rapidly that another and larger lock is now required, and work has been commenced by the

Government upon a structure which is to be 800 feet long by 100 feet wide, with 21 feet of water on the sills. This demonstrates a wonderful growth of lake traffic, which may be appreciated from the following figures of the number of vessels passed, their tonnage and the amount of some of the prominent materials transported in 1880 (the year that the Institute passed through the canal) and in subsequent years. The statistical year closes June 30th.

Year.	Tonnage actual freight.	Vessels passed locks.	Coal. Net tons.	Pig and manufactured iron. Net tons.	Copper. Net tons.	Iron ore. Net tons.	Silver Ore. Net tons.
1880*	1,734,800	3503	170,501	46,791	21,753	677,073	66
1881	1,567,741	4004	295,647	87,830	29,488	748,131	0
1882	2,029,521	4774	430,184	92,870	25,409	987,060	22
1883	2,267,005	4315	714,444	109,910	31,024	791,732	814
1884	2,874,557	5689	706,379	72,428	36,062	1,136,071	9,731
1885	3,256,628	5380	894,991	60,842	31,927	1,235,122	3,669
1886	4,527,759	7424	1,009,999	115,208	38,627	2,087,809	2,009
1887	5,900,000†						

Of the balance of the tonnage the most important factors are grain, of which 2,364,000 bushels passed in 1880 but was increased in 1886 to 19,706,858 bushels; and lumber, of which 138,688,000 feet, B.M., went through the canal in 1886, against 44,539,000 feet, B.M., in 1880.

To demonstrate the importance of the St. Mary's ship-canal a statement prepared by Mr. W. F. Phelps, from which the above table was copied, exhibits the fact that during the season of navigation the tonnage through this outlet for Lake Superior commerce is greater than that through the Suez canal, and records of July, 1886, show the following data:

Tons passed through the Suez canal, ...	528,671
Tons passed through the St. Mary's canal, ...	561,228

* Registered tonnage only was recorded in 1880.

† This estimate was made by Mr. W. F. Phelps, who also states that the greatest freight tonnage passed in one day [1887] was 50,729 tons, which was carried by 71 vessels. On the 3d of June, 1887, 49 vessels passed the canal, carrying 49,258 tons of freight, or an average of a little more than 1000 tons per vessel. The estimate of the ultimate capacity of the present lock is 96 vessels in 24 hours, or one vessel for each 15 minutes. In one day during June, 1887, 84 vessels were passed.

Since this paper was prepared the report of the Canal Superintendent has been accessible, which shows that the freight passing through the locks was in

May, 1887, ...	612,543 tons.
June, 1887, ...	953,921 tons.
July, 1887, ...	968,729 tons.

In 1886 the traffic of the Suez Canal was 8,180,000 metric tons, and the directors of the enterprise consider it almost certain that in a few years the traffic of 10,000,000 tons indicated by the International Commission as the maximum the canal could conveniently carry in the condition it was at that time will be reached.

The Dominion Parliament has appropriated $1,000,000 as the first instalment towards the construction of another ship-canal on the Canadian side of the Sault Ste Marie, connecting the waters of Lakes Superior, Michigan and Huron, which is to be constructed of as great capacity as the one now being built by the United States Government.

Our Duluth friends can demonstrate that about 90 per cent. of the Lake Superior grain-shipments of 1886 and 1887 passed through their elevators, which have received over 300,000 bushels and sent forward over half a million bushels of wheat in one day, and that additions in construction to their present storage-capacity of 11,000,000 bushels will augment it this year to 19,000,000 bushels (equivalent to one-fourth of the present annual wheat-harvest of Minnesota and Dakota), making the prospects of increased shipments from this port and through the ship-canal probable. They also point to the large amount of lumber and building-stone shipped from, and iron manufactures received at Duluth, as an evidence that industries would be well sustained here. But next to the preëminence as a shipper of wheat, Duluth's pride in lake traffic is the heavy invoices of coal which cheap back-freights make possible, the receipts of coal at Duluth having risen from 163,000 net tons in 1880 to 736,000 net tons in 1886.

The receipts of coal in the harbor during the year 1887 are estimated to exceed 1,000,000 tons; and a trip over the eight square miles of safe anchorage and dock, confined by the long arm of Minnesota Point, will exhibit what large expenditures have been made in constructing the necessary equipment for handling and room for storing coal and grain. The present aggregate capacity of the coal-docks in the harbor is given as 745,000 tons. During this inspection of the harbor, a point of considerable interest will be the construction of the grain-elevators; all of the buildings and machinery rest upon piles, and are formed of a series of walls of plank laid so as to break joint. In fact, the use of piles is more general here than it would be elsewhere, owing to the fact that a pile 25 feet long is furnished, driven and cut off below water-level for three dollars.

Interest will also be taken in the two outlets to the harbor and the piers protecting them. The one at the Superior end of the bay is the natural outlet for the St. Louis River discharge. The one at Duluth is an artificial canal, cut through in the face of serious opposition and during troublesome litigation, and was expected to exert a damaging influence on the natural outlet by diverting the current through this artificial channel. So firm was the conviction that the Duluth channel would cause the destruction of the natural outlet, that a dyke was built across Superior Bay; but as experience proved that there was an ebb and flow through the canal owing to the direction or intensity of the wind, the dyke was removed, and now two outlets, seven miles apart, each provided with its own lighthouse and equipment, give convenient access to this magnificent harbor, which already possesses elevator storage for the wheat garnered from 1500 square miles of land.

The formation of Minnesota Point is of interest as showing how, from possibly an obstruction of minor importance, this slender spit of gravel and sand has gradually lengthened, until it now sweeps in a graceful curve for nearly seven miles, the average width being but 600 feet, while in places there is but 200 feet of sand to act as a barrier to the storms which sweep over the great lake. In only one place has any artificial protection been necessary.

The visit of the American Institute of Mining Engineers in 1880 to the eastern portion of Lake Superior, and the present meeting at the western extremity, or, as it is generally called, "the head of Lake Superior," will undoubtedly bring to the members the conviction that the selection of the name "Superior" was happy, both as to the size of the lake and as to the resources of the surrounding territory. The appellation is surely a very proper one to give to an inland sea 375 miles long, 150 miles wide, with an area of 32,000 square miles and a shore-line over 1000 miles in length, placed at an elevation of 601.78 feet above mean tide at New York, but of such depth that much of the lake-bottom is below the ocean-level, numerous soundings of over 1000 feet having been taken.

That the name is equally appropriate as referring to the natural wealth and the beautiful surrounding scenery we were made aware in part during the meeting of 1880; but the knowledge of what has been accomplished in the older mining districts during the interval and the opportunity of seeing the development in the newer sections will emphasize to our minds that Lake Superior is well named.

To illustrate the geographical position of the prominent mining developments of the Lake Superior district, the annexed map (Plate I) has been prepared. In it no effort has been made to locate undeveloped properties; and the discoveries of ores in the Dominion of Canada, some of which are mentioned later, are not shown. The only reference to ore-properties which are not opened as mines and shipping ore are the xx marks east of the Vermilion Lake in Minnesota, which are introduced to show the general strike of the ore in that region.

The shipping-ports for the iron-ores and their present railroad connections are indicated. Other lines are building or projected, but as they could not all be shown correctly, the completed roads only are represented.

Marquette acts as a shipping-port for part of the iron-ores from the Marquette range, and St. Ignace to the east (not shown on the map) also receives some ore. Escanaba ships the ore from the Menominee range, also part of that from the Marquette range, and by the time this paper is in the hands of the members, railroad connection will be made so that this port can handle some of the ores from the Gogebic range. A new shipping port is being fitted with docks, etc., at Gladstone, Michigan, on Little Bay de Noquette north of Escanaba. There are docks also at L'Anse at the head of Keweenaw Bay which are used to a limited extent.

Ashland is the natural shipping port for the Gogebic ores; and those from the Vermilion district are shipped at Two Harbors. It is expected that future developments in Minnesota may encourage additional railroad facilities delivering ore to Duluth or to Grand Marais, a harbor N.E. of Two Harbors.

Menominee Iron Range.

In 1880 the Menominee iron-district was a marvel on account of its phenomenal development, and we were filled with amazement at the large mining industry which had grown in three years in what was, in 1877, an unbroken wilderness; the output of the district being as follows:

1877, . .	4,593 gross tons.	1879, . .	245,672 gross tons.
1878, . .	78,028 gross tons.	1880, . .	524,737 gross tons.

But in 1882 this district reached a maximum output of 1,135,018 gross tons, and the product of the mines of the Menominee Range in 1886 was 880,006 gross tons, making the total output 6,196,687

gross tons. The shipments of 1887 will bring the aggregate production to about seven and a quarter millions gross tons. Of this amount 49.5 per cent. was taken from three mines which are now operating. The maximum yearly product of these large mines, the date of their first shipment, and their total shipments of iron to the close of 1886 are as follows:

Mine.	First Shipment in.	Maximum Yearly Output. Gross Tons.	Total Product to close of 1886. Gross Tons.
Chapin,	1880	290,972 in 1884	1,329,686
Norway,	1878	198,165 in 1880	919,749
Vulcan,	1877	143,930 in 1886	816,305

Other mines in this district which have achieved prominence as large producers are:

Mine.	Opened in.	Has Produced to close of 1886. Gross Tons.
Ludington,	1880	466,787
Commonwealth,	1880	351,673
Perkins,	1879	342,224
Florence,	1880	323,241
Iron River,	1882	316,352
Quinessee,	1878	274,489
Cyclops,	1878	231,345

Marquette Iron Range.

In the Marquette region we were charmed with the magnificent development of the mines which in 1880 produced 1,383,910 gross tons of iron-ore, making the total for the district since its first exploitation in 1854 to the close of 1880, 14,025,740 gross tons. This district reached a maximum annual output of 1,831,357 gross tons in 1882, and in 1886 produced 1,621,887 gross tons, bringing the total up to the close of 1886 to 23,346,819 gross tons. Of this total 64.2 per cent. was contributed by six mines, all but one of which is now a producer. Their names, date of maximum output and total shipments to close of 1886 are given below:

Mine.	First Shipment in.	Maximum Output. Gross Tons.	Total Product to close of 1886. Gross Tons.
Lake Superior,	1858	296,509 in 1882	3,892,195
Cleveland,	1854	225,674 in 1884	3,497,213
Jackson,	1856	130,524 in 1868	2,604,551
Republic,	1872	277,757 in 1884	2,582 826
Champion,	1868	210,180 in 1884	1,761,564
New York,	1864	94,809 in 1870	1,035,001

No other mine in the Marquette district had produced an aggregate of 1,000,000 gross tons up to the close of 1886, but the following have reached a total output of 300,000 tons or over:

Mine.	Opened in.	Has Produced to close of 1886. Gross Tons.
Lake Angeline,	1864	896,981
Barnum,	1868	761,343
Humboldt,	1865	626,561
Michigamme,	1872	616,590
Winthrop,	1870	515,248
Saginaw,	1872	447,104
Salisbury,	1872	399,414
Pittsburgh and L. S.,	1871	365,460
Cambria,	1874	337,078
McComber,	1867	307,427

The shipments of 1887 will bring the total output of the mines of the Marquette Range to twenty-five millions gross tons.

No analyses of Menominee or Marquette ores are presented, as these appear in our *Transactions*.

Growth of the Iron-mining Industry.

It will be noted that in the seven years since the first visit of the American Institute of Mining Engineers to Lake Superior the quantity of ore shipped from the Marquette region was two-thirds as much as had been taken out prior to 1880.

Until 1884 the Marquette and Menominee ranges produced all of the iron ores sent from the Lake Superior region, but in 1885 the Vermilion and Gogebic districts were opened. As further notice of these will be made hereafter, it will only be necessary at present to mention the total product to close of 1886, viz.:

	Gross Tons.
Marquette District,	23,346,819
Menominee District,	6,196,687
Gogebic District,	877,069
Vermilion District,	592,004
To this should be added from abandoned and miscellaneous mines (not included in the above),	48,432
Making a grand total up to the close of 1886 of,	31,061,011

The output of 1887 will increase this total to over thirty-five millions gross tons.

Comparisons have often been made of the Lake Superior iron-mines and those of the Bilbao district of Spain. As the report of the Chief Mining Engineer of the Province has lately been published, showing the amount of ore produced from the Bilbao mines

in each year since 1860, the figures given are presented with contemporaneous data from the Lake Superior mines: *

Year.	Bilbao District. Metric Tons.	Lake Superior District. Gross Tons.
1860,	69,816	114,401
1861,	54,869	49,909
1862,	70,460	124,169
1863,	70,720	203,055
1864,	120,470	243,127
1865,	102,360	236,208
1866,	89,912	278,796
1867,	136,075	473,567
1868,	154,120	491,449
1869,	164,800	617,444
1870,	250,337	830,940
1871,	403,142	779,607
1872,	402,000	900,901
1873,	365,340	1,162,458
1874, Carlist War	10,821	919,557
1875, Carlist War	34,296	891,257
1876,	432,418	992,764
1877,	1,040,264	1,014,687
1878,	1,305,656	1,111,110
1879,	1.262,671	1,375,691
1880,	2,683,627	1,908,647
1881,	2,620,626	2,314,502
1882,	3,855,000†	2,966,375
1883,	3,627,752	2,341,227
1884,	3,216,321	2,518,048
1885,	3,311,419	2,456,548
1886,	3,185,228	3,562,570†
Total from 1860 to 1886,	29,040,520	30,879,014

Plate II. has been prepared to illustrate graphically the annual production of the two districts for the years given in the table. This shows that from the year 1862 up to 1876 the Lake Superior district was a larger producer than the Bilbao district, but from 1880 to 1885 the Bilbao district was the heaviest producer—the Lake Superior district coming to the front in 1886. It is probable that 1887 will show a larger output from the Lake Superior district than the maximum production of the Bilbao district.

As the difference between the metric ton and the gross ton is less than 36 pounds, it was not considered essential to bring the figures

* Prior to 1860, 181,997 gross tons were taken from the Marquette mines. The output of the Bilbao mines previous to 1860 is not known.

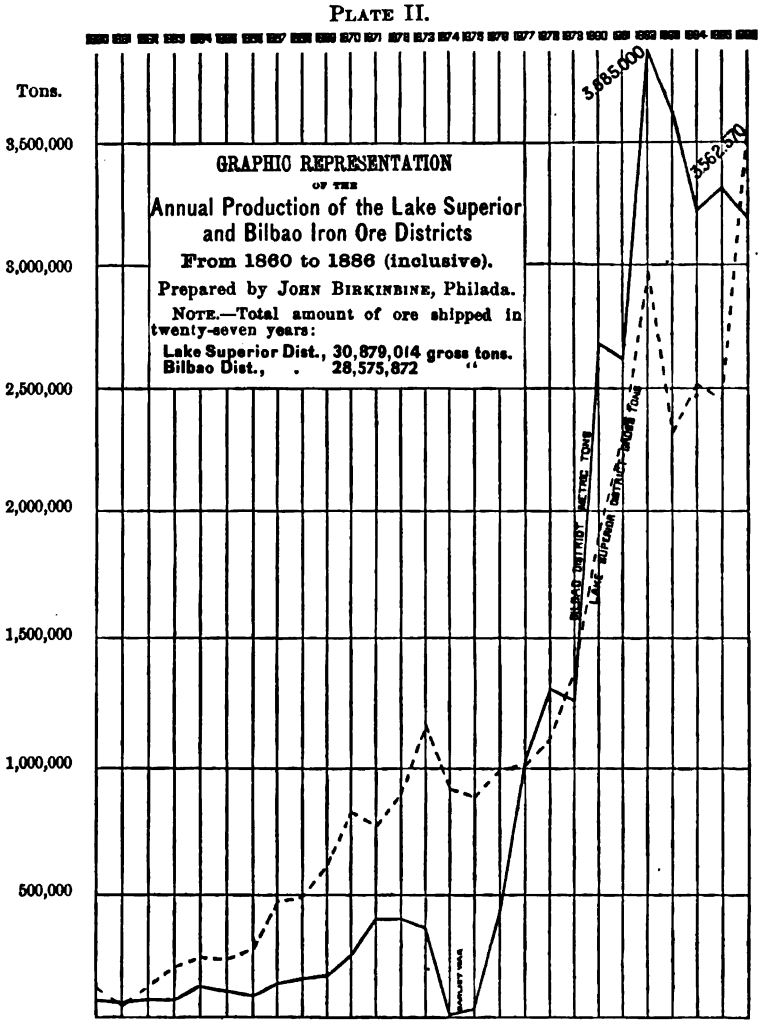
† Maxima.

to absolutely equal bases, but the total production for 27 years (1860 to 1886) may be expressed as:

	Bilbao District. Gross Tons.	Lake Superior District. Gross Tons.
Total, ...	28,575,872	30,879,014
Yearly average, . . .	1,058,366	1,143,667

Concerning the Bilbao mines, M. Alexandre Pourcel says: "In

PLATE II.



1882 the quantity of ore shipped rose suddenly to 3,855,000 tons, in order to meet an export demand of 3,692,000 tons. Since 1882 the output has steadily diminished. It may possibly rise by a few

hundred thousand tons in 1887, but it will probably remain below the maximum of 1882. The majority of the mines worked by private parties, which are specially the source of the ore exported, have been exhausted or are about to be. If the demand required it, which is not very probable, 4,000,000 tons might be mined in 1888, and as much in 1889; but it would be difficult to continue that, and it would drop to 3,000,000 tons, which is the maximum which the present state of the mines can sustain for more than four or five years. It may be foreseen, therefore, that after 1890 or 1891 the quantity of ore available for the open market will be more and more restricted, and that, as a consequence thereof, before ten years have elapsed, the output will still drop to a figure close upon 2,000,000 tons."

From the figures above given it appears that the amounts of iron-ore extracted from the Bilbao and Lake Superior districts are nearly the same, and if the output of the newer ranges, viz., Vermilion and Gogebic, are excluded from the Lake Superior quota, the comparison is still closer; the Marquette and Menominee ranges of the eastern portion of upper Michigan and Wisconsin peninsula having shipped 29,409,941 gross tons from 1860 to 1886, against 28,575,872 gross tons in the same time taken from the Bilbao mines.

We thus see that these two great iron-producing districts have in a quarter of a century produced very nearly the same quantity of iron-ores; but the Lake Superior ores will average more iron than those obtained from the Bilbao district. Whether the similarity between the output of the two regions will continue in the future is a most interesting question, and one which will bear a more elaborate discussion than can now be given to it. In the Marquette district no large mine has been discovered since the Republic was opened in 1872, although explorations have been constantly active in that district. The larger producers have a considerable amount of ore in sight, but each year apparently depletes them more and more. Lenses fail and others come in, but the workings are constantly growing deeper or more uncertain, and the difficulties and expenses of getting the ore increase correspondingly. The Champion and Republic are now down from 600 to 700 feet in depth, and those in the neighborhood of Ishpeming and Negaunee are from 450 to 550 feet in depth. In the Menominee district most of the great producers are able to maintain their positions, but both this range and the Marquette range will hardly exceed their maximum output of 1882, and may be considered at present as being worked up to their

practical capacity. We must therefore look to the newer discoveries in the Gogebic and Vermilion ranges to maintain the supremacy of Lake Superior as the great iron-producing region, and it is highly probable that new discoveries will in the near future make it possible to largely augment the product from this region. Brief descriptions of these new ranges are offered to give an idea as to their character and extent.

These remarks concerning the output of the Lake Superior iron-ore mines are not intended as prophesying speedy exhaustion; on the contrary the history of the past gives reason for faith in the future, and the prospects for some of the mines which have made notable records continuing to be large producers are favorable; while a succession of large lenses of ore may bring some mines now unimportant into prominent places as producers. But should the older districts become practically exhausted, their history would be unlike that of the Bilbao mines, for the former have largely contributed to build up a magnificent iron industry which has enriched our country, made us self-reliant, given employment to thousands of citizens; and to the Marquette and Menominee ores we owe in great part the high position which we have attained in practical metallurgy —for no nation can boast of better records from its blast-furnaces and steel-works. The Bilbao ores, on the other hand, have been chiefly used to enrich other nations, and Spain has only profited to the extent of the labor employed in the mines and by an export duty upon ore shipped. Great Britain owes much of its preëminence as a producer of iron and steel to the Cleveland ores; but relatively the ores which have been taken from the Lake Superior mines have exerted a greater influence upon iron and steel manufacture than the noted English deposit; and the high reputation gained for Scandinavian iron by the employment of excellent Swedish ores is fully equalled by the results obtained in America by the utilization of the iron-ores mined in the Lake Superior district.

Two great iron-ore districts have lately jumped into prominence under widely dissimilar conditions, at a time when their accession to the rank of sources of supply of first magnitude must be regarded as almost providential. It is not difficult to picture the results which would have followed the sudden increase in the demand for iron and for pig acceptable for Bessemer purposes particularly, had the plants of Western Pennsylvania, Ohio, Illinois and Wisconsin been forced to depend upon the two older districts, the Marquette and Menominee ranges, alone for ore. Conceding that under great

pressure they might have reached 2,750,000 tons in 1886, there would have been a deficiency of more than 750,000 gross tons. To cover this we would have been forced to draw largely on foreign sources, and finding them inadequate, would have been driven to importing enormously large quantities of pig-iron, blooms, billets, slabs, rods and rails.

A long pause followed in the development of the Marquette and Menominee regions, and American capital began to look in the direction of Canada for new sources of Bessemer ore-supply, when almost simultaneously, in 1884, the Gogebic and the Vermilion ranges sent their first cargoes of ore to market.

By the opening of the new districts the question of a full ore-supply for years to come is secured to the entire iron and steel industry west of the Allegheny mountains and probably further east.

Vermilion Iron-Ore Mines.

Minnesota is popularly known as the Gopher State, but can scarcely be credited with having earned the appellation until active exploitation brought into notice the iron-ore deposit in the vicinity of Vermilion Lake, which has since been developed so successfully that a wilderness has, in four years, been transformed into an active mining centre. Although the existence of the deposit of ore had been reported for about twenty years, no active efforts looking towards its utilization were attempted until 1883, when the first real work was done. Now, after a lapse of four years, there are excavations from which over 850,000 gross tons of iron-ore have been taken; a rail-road constructed, extending through the forest for 70 miles to Two Harbors and continuing 26 miles further along the lake shore to the City of Duluth; and over 5000 people living around the mines and dependent upon the 1300 men engaged in extracting the ore from them.

The first shipment of ore from Two Harbors was made on August 1st, 1884, but before the close of navigation of that year, 62,124 gross tons had been sent forward by lake vessels to various blast-furnaces. That the quality of Vermilion ore sustained the reported analyses, is evident from the fact that the shipments in 1885 more than trebled those of 1884, the gross amount for 1885 reaching 225,484 gross tons. In 1886 this was still further increased and 304,396 gross tons were shipped. Contracts for freight covering 400,000 tons have been made for the present year; and by the close

of navigation, an aggregate of over 1,000,000 tons will have been taken out of the mines in the vicinity of Tower, all of which yielded 62, and much of it 68, per cent. of iron, and nearly all of it shipped has been within the Bessemer limit of phosphorus.

A personal examination of all the analyses (115 in number) taken from a stock-pile of 165,000 tons showed an average of iron 67.7; phosphorus, 0.06; silica, 1.5. The range of the entire series of analyses was found to be iron, 65.29 to 69.28; phosphorus, 0.049 to 0.067; silica, 0.68 to 3.79. Several analyses show over 69 per cent. of iron, indicating that the ore closely approximates the theoretical purity of red hematite. The extreme range of phosphorus found in analyses since the opening of the mines to date is reported as from 0.021 to 0.11, but the ore as mined shows from 0.04 to 0.07 of phosphorus. Mr. F. Prince, a member of the Institute and mining engineer for the owners of the mines given below, furnishes the appended analyses.

To reach the Vermilion iron-ore region the Duluth and Iron Range railroad skirts the northern shore of Lake Superior, from Duluth to Two Harbors, where the shipping-docks, with a capacity of 14,000 tons, and railroad settlement are placed. From Two Harbors the road follows a general northwest direction, rising in the first 3 miles 400 feet, and in the first twelve miles 1100 feet, to the summit of what might be called the coast range, then crossing the drainage area of the St. Louis river to the Messaba range, a ridge of granite, where explorations are now in progress for magnetic ores. Continuing 15 miles beyond the Messaba range the Vermilion iron-ore mines are found at an elevation of 900 feet above Lake Superior. The openings indicate a number of ore-lenses of large size lying on two nearly parallel hills, but ore has also been found in wells sunk between the two ridges, and explorations with the diamond drill are now in progress in the intermediate territory.

These lenses, as developed, are known by various names; the Breitung mine, the most westerly opening, has produced over 62,000 tons of ore; next to and adjoining this opening is the Tower No. 1, which is an excavation 250 feet in length, 180 feet maximum width, and 100 feet deep, from which nearly 215,000 tons of ore have been taken. Immediately adjoining this the Tower No. 2 mine shows a length of 230 feet, a width of 60 to 80 feet, and a depth of 100 feet; it has produced over 170,000 tons of ore. Between the Tower mines and the Ely opening there is some unexplored ground. From the Ely pit, which is 400 feet long and 50 feet wide, over 88,000

tons of ore have been taken; and immediately adjoining the Ely is the Stone pit, 700 feet long and 100 feet deep, showing a width of 50 feet, which has produced nearly 240,000 tons of ore. The most easterly excavation is the Stuntz mine, 250 feet long, and from 20 to 50 feet wide, from which about 25,000 tons have been raised. These six openings are on one ridge and cover a strike of about one mile. On a second ridge two openings known as the Lee mines have produced about 50,000 tons of ore.

All of these workings have been open quarries, but at present preparations are being made, and shafts sunk, for underground operations. The statement has been made that although the thermometer frequently indicated from 30 to 40 degrees below zero, it was not found necessary to suspend work day or night during the past winter in these open pits. Many of the houses at the mines are owned by the operators; the iron company leasing the ground for a term of years at a nominal rental, and permitting the owner of the house to dispose of it to persons in the employ of the company.

The ore being hard and dense, lying practically free from any foreign material, except the generally well-defined seams of jasper and "soap-rock," demands the use of rock-drills and high explosives; but as comparatively little must be taken out beside the ore, the cost for explosives per ton of ore is less than is generally estimated.

The product of the mines is divided into three grades, the first, called Bessemer, being guaranteed to carry not less than 66⅔ per cent. of iron, and under 0.06 per cent. of phosphorus. The second class, called Minnesota, is guaranteed to carry 66⅔ per cent. of iron, but is sold without guarantee as to phosphorus, while the Red Lake ore contains not less than 62 per cent. of iron, but without any limitations as to phosphorus. It is reported that the company's contracts for the season are as follows: 230,000 tons of Bessemer, 100,000 tons of Minnesota, and 70,000 tons of Red Lake.

The moderately fluctuating character of the deposit is well illustrated in the Stone and Ely openings, where it is seen bulging and contracting on the dip, and liable to run into jasper or be displaced by soap-rock (decomposed diorite) on the strike. With a standard of shipping-ore as high as that of the Company, any parts of the vein seamed with jasper must promptly wander on the dump; but it is probable in the future that some of this ore will be selected for use nearer the mines than the lower Lake ports, to which the ore is now sent. Close attention must also be paid to the phosphorus-content.

It needs constant watching, since it fluctuates closely above and below the Bessemer limit.

The output of the different openings together with the date on which work was begun on them to July 25th, 1887, is shown in the following table:

*Output of Vermilion Iron Range to July 25th, 1887, from Date of Opening.**

Mine.	When Opened.	Total Product.
Stuntz,	August 1st, 1884,	24,399
Stone,	May 1st, 1884,	239,866
Ely,	December 29th, 1886,	88,307
Tower, No. 1,	July 20th, 1884,	214,794
Tower, No. 2,	April 11th, 1886,	173,831
Breitung,	May 8th, 1884,	62,098
Lee,	December 13th, 1884,	50,270
Total,		853,565

The output of July will probably reach 40,000 tons, the product in the winter months of course being considerably less. During the summer as large a quantity of ore as possible is sent direct to the shipping-port. Still some stocking is unavoidable, and after the closing of navigation all the product must be stocked. This involves rehandling, which, with the cost of raising ore and waste rock, charges of explosives, drill-steel, smith's waste, fuel for engines, etc., will carry the cost up close to $2.00 per ton.

Parallel to and about 12 miles south of the Vermilion range is the Messaba range, in which a number of magnetic ore indications have been explored, but little reliable information can be secured about them. It appears that at least some of the ore is titaniferous, although none of the analyses which are given below indicate this definitely; and that a third range, closer still to the Lake, is reported even higher in that objectionable metal, to the point of being at present worthless.

Indications of ore are reported from widely separated points on the Vermilion range, and reports of discoveries come in almost daily, the woods being literally full of prospectors. In a country overgrown with timber and cut up by swamps and lakes, prospecting naturally proceeds under difficulties, and it may readily be perceived that many important deposits still remain undiscovered.

* It was not practicable to obtain the shipments of all the mines of the other districts to this late date, by reason of their control being in the hands of various parties—but their output to the close of 1886 is given.

The development of a property such as that described has naturally encouraged exploration of adjacent territory, and some very satisfactory discoveries have been made about 20 to 40 miles to the east. These will receive attention from others, but for the present some analyses of the Vermilion and other ores will be given.

Analyses of the Vermilion iron-ores made by Mr. Prince show:

Iron,	67.99	68.37	68.32
Phosphorus,	0.053	0.057	0.046
Silica,	1.35	1.10	1.35
Alumina,	undetermined	0.50	0.25
Magnesia,	undetermined	0.014	nil.
Sulphur,	0.005	0.007	nil.
Loss by ignition,	undetermined	0.56	0.66

Determinations of other Minnesota ores show:*

	1.	2.	3.	4.	5.	6.
Iron,	63.13	65.62	64.33	58.74	54.35	57.88
Phosphoric acid,	0.37	0.18	...	0.23	0.055	0.067
Water,	...	...	...	0.45	...	...
Silicious matter,	8.70	4.06	9.98	13.28	21.47	18.075
Sulphur,	nil.	...	...	nil.	nil.	nil.
Lime,	1.69	3.15	2.04	1.18	...	...
Magnesia,	0.52	0.50	1.29	0.98	...	...
Manganese,	0.54	0.42	1.01	1.36	trace.	trace.
Alumina,	0.22	0.35	...	0.62	...	...
Undetermined,	...	...	...	...	3.145	1.958
Specific gravity,	...	...	...	4.53	...	...

Nos. 1 and 2 were by Taylor of Cleveland.
No. 3 by Campbell of New York.
No. 4 by Britton of Philadelphia.
Nos. 5 and 6 by Courtis of Detroit.

The natural harbors for the shipment of the Minnesota iron-ores thus far developed are Duluth, Grand Marais and Agate Bay, all good harbors, and the last two notable for the near approach to semicircles made by the shore contours.

Grand Marais, the smaller of the two, represents, with its shore line, nearly a true semicircle with a radius of 1400 feet, the chord of this arc being in part formed by projecting rock ledges, which reduce the entrance to 1100 feet. This has been further reduced by a breakwater 350 feet long, constructed by the United States Government.†

* 1, 2, 3 and 4 are from the Messaba Range, and 5 and 6 from near Grand Marais.

† There are two Grand Marais harbors on Lake Superior, one on the north shore in Minnesota, herein described, and one on the south shore in Michigan, 80 miles east of Marquette.

Agate Bay (now recognized as Two Harbors) is larger than Grand Marais Harbor, and is not so true a semicircle, the contour being more helmet-shaped. The major diameter, 3750 feet, is open to the lake, except for 400 feet, which is closed by the breakwater constructed by the Government. By extending this breakwater from either shore, an opening of 1350 feet will be left leading into the harbor.

Gogebic Iron-Ore Range.

If the development of the Vermilion iron-ore district is startling, that of the Gogebic iron range in Wisconsin and Michigan is even more so; for although the existence of visible outcrops of ore was long known, and considerable amounts of money were expended by some of our large iron companies, no actual exploitation can be considered as having been made until the year 1885, when railroad-connections were completed to Ashland and dock-facilities provided. During 1884 1022 gross tons were sent from what is now the Gogebic region, but in 1885 this amount was increased enormously, and the shipments amounted to 119,766 gross tons; and in 1886 756,281 gross tons were sent to market. It is expected that the two railroads connecting the mines with Ashland and the three ore-docks constructed in that harbor (two of which have capacities of 25,000 tons each and one of 20,000 tons) will handle over one million gross tons of ore in 1887, and bring the total output of the mines of the Gogebic region at the close of the year to more than two million tons. This from a district which four years ago was unsettled territory without railroads or inhabitants, but which is now dotted along the strike of the ore-bearing rock with progressive towns, having an aggregate population of probably 15,000 inhabitants; and four railroads are moving for the trade.

The development along the apparent strike of the ore covers nearly twenty miles in length, and active exploration is in progress for as great a distance both east and west of this territory and also upon a parallel ridge twelve miles south, the latter being for magnetic ore. The center of the present developed iron-ore properties is near to the Montreal river, which forms the boundary between Wisconsin and Michigan; the largest producers up to the present time are, however, chiefly in the State of Michigan.

The ores in the Gogebic range differ from those of the Vermilion range in being softer, and, therefore, more easily mined, yielding less iron and also less phosphorus, but carrying a greater percentage

of manganese, and more moisture. The dip of the Vermilion ore is nearly vertical; that of the Gogebic ores approximates 70 degrees.

The ores of the Gogebic range evidently lie in a series of lenses, often connected or in echelon, and the region has attained considerable notoriety from the fact that ore-indications or actual deposits have been found upon nearly every property along the apparent strike of the vein-matter.

The general geology of the district is explained by Mr. J. Parke Channing, of Bessemer. He says: "The chief characteristic of the Gogebic range, and that which makes it so easy to explore, is the regularity and persistency of the formation and the strongly marked character of the foot-wall, which dips from 45 to 70 degrees to the north, being flattest near Sunday Lake. The sinking of a shaft on the North Aurora, which it is thought will reach the continuation on the Aurora vein at a depth of about 1200 feet, will illustrate the faith in its persistency.

"Their magnitude (aside from the occurrences of horses and fluctuations in their width) has been affected in at least one instance by a dyke which cut clear across the vein. In a few cases dykes of this description have been sunk through, and the ore found under them. Faults of the entire formation have been suspected in one or two localities, but are not directly proven."

The apparent extent and continuity of the deposit has encouraged the formation of a large number of mining companies, many of which must either be consolidated or stand the test of the survival of the fittest. There are, however, some truly remarkable openings along the range, and notwithstanding the apparent continuity of the ore-body, it is probable that the bulk of the shipments from the Gogebic Range will, as in the Marquette and Menominee Ranges, come from a few large mines. Of the total amount of ore shipped to the close of 1886, namely, 877,069 gross tons,

The Colby mine supplied,	342,766	gross tons.
The Norrie mine "	140,263	"
The Aurora mine "	98,802	"
The Ashland mine "	80,486	"
Making a total of,	662,317	gross tons,

or 75.7 per cent. of all the ore shipped up to the close of 1886. These four mines will be the largest producers in 1887, but several others will be added, which may equal the output of the Aurora mine in 1886.

Most of the ore is won by underground mining, the deepest excavation being at present in the Ashland and Norrie mines, where a depth of 285 feet has been attained. But there are, or were, some remarkably handsome shows of ore in the open cuts; notably, at the Colby, Aurora, Iron King, Puritan, Ironton in Michigan, and the Montreal, Trimble, and Kakagon in Wisconsin. In the Colby mine, the presence of a large horse between two bodies of ore, and the difference of the amount of manganese carried by the ore on either side of the horse, encouraged the belief which the development of some other mines had given rise to, namely, that there were two distinct veins of ore in the territory. It was at this mine that the earliest exploitation was made, and from which the first ore was shipped, in 1884, namely, 1022 tons; in 1885 it increased its output to 84,312 gross tons, and in 1886 to 257,432 gross tons. This output, however, was probably affected by the fact that the mine is operated under a three-years' lease, and there is, therefore, every encouragement, particularly when prices are satisfactory, to work the deposit to its utmost.

The Norrie mine commenced operations in 1885, in an open pit, from which 15,419 gross tons were taken, but in 1886 carried its workings 200 feet underground, and shipped 124,814 gross tons. And its immediate neighbor on the east, the Aurora mine, took out 4249 tons in 1885, and 94,553 gross tons in 1886, from a magnificent open-pit working. At present, the ore taken from the Aurora mine is from underground workings.

On the west of the Norrie mine is the Ashland mine (the most westerly of the Michigan mines), which shipped 6471 gross tons in 1885, and 74,015 gross tons in 1886, from underground workings. These two mines, and some others, employed the Nevada system of timbering, but its cost will soon demand some other method of sustaining the hanging wall, and already gives rise to speculation as to the probable necessity of establishing more permanent methods of development than sinking a slope in the soft ore along the footwall, which is the custom in the Gogebic range. The caving system is being experimented with in the Ironton mine.

In Wisconsin, the Germania mine, near the town of Hurley, shows developments similar to those named; while in the mines to the west, the character of the deposits is exposed in several open pits and numerous underground workings. The range is of sufficient importance to demand a complete paper, and no detailed description will here be attempted, but the following analyses are presented to show the general composition of the ores (when dried):

	Iron King.		Norrie.	Aurora.	Ashl'd.	Germania.		
					48 cargoes.			
Iron,	60.85	55.74	63.57	62.83	62.93	64.50	59.38	59.70
Phosphorus,	0.027	0.034	0.06	0.0474	0.0278	0.047	0.058	0.056
Silica,	5.44	3.47	4.09	5.18	...	3.65	...	...
Manganese,	1.30	12.28	...	...	...	...	...	...

	Pabst.				Superior.				
Iron,	58.47	58.46	...	58.38	62.36	64.83	65.18	64.25	59.30
Phosphorus,	0.040	0.044	0.31	0.36	0.06	0.047	0.40	0.054	0.079

Colby Mine.

	North Vein [?].	South Vein [?].			
Iron,	59.50	60.00	58.61	59.30	57.736
Silica,	2.50	2.50	3.89	3.55	6.863
Phosphorus,	0.043	0.043	0.05	0.039	0.047
Manganese,	3.00	4.50	...	...	1.916

Montreal Mine.

Iron,	66.00	65.22	65.35	66.00	66.50	67.65
Silica,	1.40	3.45	2.80	1.72	1.30	1.20
Phosphorus,	0.028	0.038	0.045	0.041	0.031	0.046

The Gogebic ores carry, on an average, about 60 per cent. of iron when dry, some of the mining being above that figure. Carelessness in mining, particularly in open pits, is almost invariably followed by a decline in the iron-contents. Although the ore from a few mines is almost too high in phosphorus, the average product of the district is of Bessemer grade. The ore carries about 13 per cent. of moisture in the winter, sometimes running as high as 15 per cent., while in the summer it is from 4 to 5 per cent. less, averaging 7 to 10 per cent. The furnace-averages of a number of analyses of ore from mines which are large producers are as follows:

Mine.	Iron.	Phosphorus.	Manganese.
Anvil,	60.00	0.05	...
Ashland, 1886,	64.00	0.04 to 0.05	...
Ashland, 1887,	61.00	0.04 to 0.05	...
Aurora (open pit),	57.00	0.035 to 0.04	...
Colby,	60.00	0.045	3.00
Germania,	60.00	0.05	...
Iron King, north vein,	61.00	0.03	...
Iron King, south vein,	58.00	0.04	4.00
Ironton,	61.50	0.045	...
Kakagon,	60.00	0.05 to 0.06	...
Norrie,	62.00	0.05 to 0.06	...
Pabst,	57 to 61	0.035	1.00
Puritan,	61.25	0.045	...
Sunday Lake,	61.00	0.04	...

The days of exceptionally cheap mining in the Gogebic range are over. Large quantities of ore have undoubtedly been taken from open cuts at low figures, but by the close of the year few if any of the mines will be operated in this manner. The royalty is also a continued and heavy charge upon the profits: this is, in most cases, 50 cents per ton, the stipulation being, that at least 10,000 tons shall be mined each year from each 40 acres leased, so that the owners will receive $5000 a year, whether the mine is worked or not. This is very hard on what are merely prospects. In some cases, the selling price at Cleveland is made the base of a sliding scale.

The average cost per ton of ore is probably between $1.25 and $1.75, counting a fair amount of dead-work. An item that is constantly becoming more important is timbering, which costs probably 50 cents per ton.

The railroad-freights to Ashland, including dock-charges, are now 80 cents per ton.

From present development, it would appear that the ores of the Vermilion range can be mined as cheaply as those of the Gogebic range. The former require more labor and explosives, but the ore forms its own supports; the latter are soft, and while more easily removed, demand expensive timbering, and it may be necessary, to secure permanent work, to sink the slopes in the foot-walls. The amount of water to be raised is, with a few exceptions, remarkably small considering the territory covered by these two ranges.

Canadian Iron-ores.

The statement that new iron-ore deposits will be found adjacent to Lake Superior, is not intended to be interpreted as confined to the United States side, although it is probable that future large developments will be made there. But Canada will also add to the ore supply. As an instance, I am permitted to give the following data concerning a remarkable deposit of magnetic ore on the north side of the lake. The location known as "Ore Hill" is northeast of the Vermilion ore range of Minnesota, and 130 miles west of Fort William on Thunder Bay, where, owing to the deep water close to shore, excellent shipping facilities can be secured.

The outcrop is about thirty miles southwest of the Canadian Pacific Railway crossing of the Seine river, about 100 miles from Fort William. This deposit of ore is reported to be a high-grade Bes-

semer, the ore rising above the surrounding plain to a height of 100 feet, is in two veins combined with a parting of diorite nearly or quite 100 feet wide. The country from the Canadian Pacific main line is level, following the valley of Seine river, and this iron hill rises out of a level plain, so that a track can be laid on both sides of the hill, and the ore thrown down and loaded on cars at small cost. It is a black, magnetic ore, easy to drill and break; very little machinery would be required; no pumping or hoisting.

The following analyses made by the Union Steel Company from sampling by a Marquette mining superintendent of recognized ability, show

Sample.	Fe.	P.	SiO_2.	TiO_2.	Al_2O_3.	Mn.	S.	CaO_2.
1, . .	63.47	.035	6.60	none.	1.09	trace.	.22	1.28
2, . .	62.84	.030	7.30	none.	1.80	trace.	.08	.99
3, . .	63.97	.011	5.89	trace.	.98	trace.	.04	1.40
4, . .	70.06	.025	2.43	none.	.67	trace.	.06	...
5, . .	67.95	.005	2.63	none.	...	trace.	.05	...
6, . .	66.92	.012	...	trace.	...	...	trace.	...

This ore, owing to its high grade and low phosphorus and the exceptional character of the deposit which permits of an enormous quantity being quarried down and shipped at a low cost, will, it is believed, yield a handsome profit at Lake Erie and Lake Michigan ports, after paying a duty of 75 cents per ton.

The recent large increase in the Canadian tariff on iron to $4 per ton on pig- and $13 on bar-iron, with anthracite coal on the free list to encourage home manufactures in the Dominion, may open the way to smelt this ore at Fort William.

Copper.

The recent development of the copper-mines on the Keweenaw Peninsula has been less marked than that of the iron-ores of Lake Superior region, but notwithstanding the active competition caused by the opening of large western mines the progress has been very satisfactory and the Lake Superior mines are operated successfully, where the material extracted yields less than one per cent. of copper.

The equipments which are connected with these mines, and which in 1880 raised the ore from depths of 2500 feet, but now elevate it 3800 feet,* are marvels of American ingenuity, and the appliances

* The depths of the shafts of the Calumet and Hecla copper-mine are reported as follows: Calumet Branch, No. 1, 3900 feet; No. 2, 3800 feet; No. 3, 3400 feet;

for separating the native copper from the gangue are wonderfully complete.

In a review of the Lake Superior mining industries for 1886 Mr. C. D. Lawton, Commissioner of Mineral Statistics for the State of Michigan, says: "Copper-mining in Michigan has become a very uniform industry; prices may vary, but the work at the mines goes on steadily with little apparent change. There is all the while a gradual increase in the magnitude of the operations, resulting in an increased production and a corresponding lessening of cost. Few, even of mining men, who are not acquainted with the details of the work at our Michigan copper-mines are ready to give credence to the statements that rock which has a gross value of but $1.65 per ton, or less than 15 pounds of refined copper, on an average, to the ton, can be mined at a depth of more than 1000 feet below the surface, hoisted, broken, stamped, washed and separated, smelted, taken to market, and sold, and still leave a net profit of 22 cents on every ton. Each year, as the price of copper has diminished, the daily production has been increased in order to reduce the average cost. In 1876 the cost of stamping and washing was 88 cents per ton, while now it is but 30 cents. The total average cost per pound of copper obtained at the Atlantic mine ten years ago was 22 cents; in 1885 the same was produced for 9⅛ cents. The mine is no richer than formerly; in fact there has been practically no change in the quality of the rock. The advantage gained is due to improved facilities for mining and manipulation, to a better comprehension of the work, etc.

"No business in the land of equal magnitude is more systematically and carefully conducted than is the copper industry of Michigan. There is none more legitimate, or that is conducted with more freedom from speculation and from those manipulations of stock which unfortunately too frequently characterize mining."

Up to the close of 1880 the copper-mines of the upper peninsula of Michigan had produced a total of 268,676 gross tons, and in the six years since that date 175,610 gross tons have been added to the output, making a grand total to the close of 1886 of 444,286 gross tons of ingot-copper.

The figures for the various important mines are not at hand for 1880, but comparing those for 1881 and 1886 we have the following:

No. 4, 3800 feet; No. 5, 3400 feet: Hecla No. 1, 3800 feet; No. 2, 3800 feet; No. 3, 3300 feet; No. 4, 900 feet; No. 10, 800 feet; No. 11, 1200 feet; No. 12, 1100 feet.

	1881. Gross tons.	1886. Gross tons.
Calumet and Hecla,	14,000	22,553
Quincy,	2,458	2,629
Franklin,	1,196	1,904
Osceola,	1,866	1,589
Atlantic,	1,129	1,564
Central,	633	1,122
Huron,	114	890
Allouez,	658	770
Copper Falls,	299	625

In the *Mineral Resources of the United States for* 1885, p. 210, a table is given showing that in 1845, when the Lake Superior District produced 12 gross tons, the total production of the United States was but 100 gross tons of copper. In 1856, out of 4000 gross tons of copper, the Lake Superior District produced 91.6 per cent., and in 1865, when the product of the United States was 8500 gross tons, that of the Lake Superior District was 6410 gross tons, while in 1871 the total copper output was 13,000 gross tons, the Lake Superior District producing 92 per cent. of this amount. Since that date the proportion of the copper production of the Lake Superior District, as compared with the total output of the United States, has declined, but in 1886, out of 71,980 gross tons, this region contributed 36,033 gross tons, or 50.05 per cent.

From the figures above given we see that in the six years since 1880 the Lake Superior copper-mines have produced 65.4 per cent. as much as they did from 1845 to the close of 1880. In the above list of large producers the Calumet and Hecla leads all the others, and in 1886 produced 22,553 gross tons, its maximum annual output. In 1867 this mine produced 603 gross tons of copper, and its output in 1870 rose to 6277 gross tons; it passed 10,000 gross tons in 1877, and has, up to the close of 1886, produced a total of 217,547 gross tons of copper, or 57.8 per cent. of the output of the Lake Superior Region from 1867 to 1886, both inclusive.

Precious Metals.

For several years exploitations for precious metals have been carried on in the Marquette region, and late discoveries are reported there of gold-ore, unusually rich. Reports of a similar character come from Northern Minnesota.

The development of Silver Islet on the Canadian shore, which has been described in our *Transactions*, and the late finds of silver-ore reported at Thunder Bay, attract attention to the region.

These facts, together with specimens obtained in various localities, lead to the belief that gold and silver in workable amounts may be added to the products of this remarkable region that has already contributed so much mineral wealth from a territory of which but a limited portion has been practically explored. As the lumber industry denudes the forest areas the opportunities for explorations are increased and the exposures of mineral wealth are more readily traced, so that, in a section of which so much is still a wilderness, there is, undoubtedly, other natural wealth to be exposed. There is, however, an element of future danger in the too rapid denudation of the forest areas without provision for re-forestation in a region lying between great lakes which may be storm-swept or subjected to severe climatic changes.

Building-stone.

The abundance of good building material must also command attention, and opportunities will be offered for an examination of the granite quarries of the Messaba Range, the sandstone quarries at Fond du Lac, and the slates at Thomson, all on the drainage area of the St. Louis river, the most important tributary of Lake Superior.

St. Louis River.

The Dalles of the St. Louis river form an attractive feature of the meeting, and admiration of a torrent of water representing the volume gathered from nearly 4000 square miles of drainage area, dashing over cataracts and rapids on its way to feed Lake Superior, may be supplemented by estimates of the commercial value of the water power which could be made available for various purposes.

In passing from Duluth to the Vermilion Lake iron district, the Messaba range must be crossed. This range is the divide between the waters flowing into Lake Superior and those which find their way into the Mississippi river or into Hudson Bay. The country where the St. Louis river and other streams have their origin, consists of a series of swamps separated by ridges; and in a limited area it is possible to follow the outlet from nearly adjacent swamps until they contribute their waters to streams, which finally reach the ocean through Hudson Bay, St. Lawrence Bay, or the Delta of the Mississippi, and yet this gathering ground is but 1600 to 2000 feet above the ocean level.*

* Mr. R. H. Lee, general manager of the Duluth and Iron Range railroad, states that this road crosses, at an elevation of 1597 feet above the ocean, the divide

This comparatively low, continental divide, which separates the waters which flow north and south, is an interesting topographical feature, the study of which, in connection with the evidences of glacial wearing and the presence of drift, is most instructive.

The St. Louis river may be estimated as 100 miles in length, with head-waters 1000 feet above Lake Superior, but more than half of the fall occurs in the latter part of its flow. From Knife Falls to Fond du Lac (a distance of 13 miles) the fall is 576 feet, and from Fond du Lac to Duluth (15 miles) there is no fall, the river being navigable by a natural channel.

The portion of the river visited by the Institute is the seven miles from Thomson to Fond du Lac, in which the water falls 480 feet to lake-level, and the navigable portion of the stream between Fond du Lac and Duluth.

The United States government has been for years collecting data concerning the rain-fall upon the drainage of the upper Mississippi, and finds that, with an average annual rain-fall of 30 inches, an ordinary minimum flow of 0.2 cubic feet per second for each of the 20,000 square miles of territory drained passes the St. Anthony's Falls at Minneapolis. The average rain-fall at Duluth for twelve years has been 34 inches, of which over 73 per cent. has occurred between May and October. As the length of the flow is less on the St. Louis river to Fond du Lac, than of the Mississippi river to Minneapolis, and the rain-fall greater, a minimum flow of 0.3 cubic feet per second for each square mile of territory drained is assumed.

Therefore, using Fairbairn's formula, viz., 12 cubic feet of water falling one foot per second equal one horse-power, we have: Area drained, $\frac{4000 \text{ square miles} \times 0.3}{12} = 100$ horse-power per foot of fall available.

But the average flow will be fully double the minimum, so that in estimating for the utilization of the power of the St. Louis river, during the season of navigation provision could be made for the application of 150 horse-power for each foot of fall available. Much of the descent of the river can be cheaply developed; and, if but 400 of the 480 feet of fall between Thomson and Fond du Lac is utilized, there would be a force available of 60,000 horse-power. The power which could be developed at Fond du Lac (estimated

between the waters of Vermilion Lake which flow into Hudson's Bay, and those passing down the St. Louis river into Lake Superior. This divide is 59¼ miles north of Two Harbors, or 85¼ miles from Duluth.

at 5000 horse-power) would be so located that vessels could come to the site of the mills or factories for their load of flour or other manufactures.

The possible means of utilizing the power embrace various forms of motors and also devices for conveying power by wire-rope transmission, by compressed air and by electricity. The river, which in several instances produces within short distances falls of greater altitude (in one case 105 feet) than is generally considered desirable for operating turbine wheels of large diameters, owing to the high velocity at which they must run, seems to present for consideration the problem of transmission of power by compressed air. The operation of large mining plants in the Lake Superior region by air carried under pressure for miles, would seem to offer possibilities of, in this way, transporting at least some of the horse-power of the St. Louis river to Duluth or intermediate points. The fall which can be secured at several points, suggests that its force could be applied direct without the intervention of wheels, gearing, etc., by combining the features of a water-pressure engine and air-compressors.

What more picturesque journey can be described than one which would be made by a log starting near the low continental divide above referred to at the head-waters of the St. Louis river, at an elevation of 1800 feet above the sea, down this river and through the Dalles out into the calm waters of St. Louis Bay, and then with an average volume of 2500 cubic feet per second through Superior Bay, where, joining the waters of the Nemadji river, it passes into the Great Lake, whose elevation is 601.78 feet, which it traverses for 375 miles, leaving it with a volume of 90,783 cubic feet of water per second pouring over the Sault Ste. Marie; and then through the St. Mary's river into Lake Huron (elevation 581.28 feet); thence through St. Clair Lake and river with a volume of 234,000 cubic feet per second to the Detroit river and Lake Erie (elevation, 572.86), from which it emerges through the Niagara river with a volume of 243,000 cubic feet per second, and dashes over the rapids and cataract to Lake Ontario, elevation, 246.61 feet? Traversing, this the last lake of the great chain, it completes the journey of 1500 miles through them and the St. Lawrence river, rushing past Ogdensburg with a volume of 325,000 cubic feet per second and descending, often by rapids, 1100 miles further to the ocean with an immensely augmented volume through the St. Lawrence Bay. A truly picturesque but hazardous journey.

Manufactories.

As some of the features of the head of Lake Superior have been presented, the thought is naturally suggested that with the abundance of high-grade iron-ores accessible there would appear to be good opportunity for their smelting, and that the water-power of the St. Louis river if utilized could sustain important manufacturing establishments. A brief discussion of the possibilities of developing iron and other industries will therefore be in place.

Superior Bay or Duluth Harbor is located nearly midway between the Vermilion and Gogebic ore-ranges; the Tower mines of the former being 96 miles, and Hurley, the business centre of the latter, 110 miles by rail from Duluth. The shipping-ports are, however, nearer, and in discussing the ore-supply it is but fair to consider the ore delivered at the same points as for shipment elsewhere, *i.e.*, the Vermilion ores at Two Harbors, 26 miles, and the Gogebic ores at Ashland, 70 miles from Duluth. At each of these points extensive ore-docks are provided for the reception and shipment of ore by boats, and for the use of these docks a charge of probably 25 cents per ton will be made. If then a railroad freight-rate of say one cent per ton-mile is allowed, the Vermilion ore could be hauled to Duluth for practically the same charge as is made for the ore in pockets at the dock of Two Harbors, and 50 cents per ton advance over the shipping rate for Gogebic ores at Ashland should deliver them at Duluth. There would be a shifting and unloading charge at Duluth which may be taken at 20 cents per ton; but the advantage of continuous shipments during the whole year would probably permit rates for ore which would make this charge practically *nil*, and it does not appear improper to assume for an estimate of the cost of manufacturing pig-iron, that the Vermilion ores can be supplied the year round to a smelting-works on Superior Bay at the same average charge as is made during the shipping-season for ore delivered on board vessels at Two Harbors, and that the Gogebic ores could be furnished to a plant at Duluth for 50 cents above the season's shipping-rate. In other words, so far as an ore-supply is concerned, a plant on Superior Bay would have an advantage over one at Cleveland or Chicago of the freight-charges from Two Harbors, now $2.00 to 2.50 per ton, but probably an average of $1.50 per ton to Cleveland for a term of years; and the difference in the cost of the Gogebic ores would be 50 cents less than the above, or probably an average of $1.00 per ton, below their cost at Cleveland.

By using the leaner ores, which will not so readily stand shipment

over long distances, a further reduction may be secured; but this naturally becomes a subject for detailed estimates of actual operation.

Flux may be conveniently obtained, but as its quality is undetermined, all calculations can with propriety be made upon the use of limestone from the Lake Erie islands, which has been well tried and approved as a flux for blast-furnaces, and of which large quantities are now brought to Duluth to be burned to lime, at a cost laid down on the wharf of $1.50 per net ton.

The problem of cheap manufacture, however, is dependent on the fuel-supply, and as no mineral fuel is obtained convenient to Duluth a smelting-plant must necessarily be operated with charcoal, or mineral fuel must be brought to Duluth.* An attempt to produce charcoal-iron has been made at Duluth, but as the ores were then brought from Marquette (the Vermilion and Gogebic ores being inaccessible at that time), and as the works did not embrace some of the later more economical features, the inactivity of the Duluth blast-furnace (particularly when the legal complications which exist are considered) does not augur unfavorably for the smelting of iron on Lake Superior with charcoal obtained from woodlands conveniently reached by rail or by water-carriage, or for the application, to manufacturing purposes, of gaseous fuel, made, as in Sweden, from the sawdust and mill-refuse which abounds. A charcoal blast-furnace is now being constructed at Ashland, Wisconsin, to smelt the Gogebic ore from fuel produced in the timber-lands convenient to that locality; and a plant at Duluth would be as favorably located as at Ashland.

As already mentioned, the receipts of coal at the docks of Superior Bay will probably exceed one million tons in 1887, and this large amount has only been handled because of the market available and the possibility of cheap transportation. The heavy shipments of flour, wheat and other grain from Duluth harbor, supplemented by numerous cargoes of ore sent eastward from near the head of Lake Superior, furnish an excess of outgoing freights which permits cheap western rates on coal and other heavy material, and have given Duluth an advantage over other lake ports which has done much to build up the large coal trade. The rapidity with which the Northwest is being developed, encourages faith in still greater shipments of cereals, and the inspection by the Institute of the ore-regions will demonstrate the fact that the mineral output is destined

* Extensive beds of peat are reported as existing in the swamps which cover much of the area within 100 miles of Duluth.

to be greatly augmented. These facts will prove that the opportunities for low rates on coal will be continued, and a permanent supply can be depended upon. As showing the relative advantages of various ports, the following ranges of vessel-freight-rates in 1886 on anthracite coal are given.

From Buffalo to Duluth, . . .	35 cents to $1.00 per net ton.
From Buffalo to Milwaukee, . .	45 cents to $1.40 per net ton.
From Buffalo to Chicago, . . .	50 cents to $1.40 per net ton.

These are taken from the *Coal Trade Review*, and represent the maxima and minima of the season's shipment.

The price of anthracite coal at Buffalo in 1886 was from $3.57 to $4.15 per ton.	
To the larger figure we may add for dock charges and freight, .	1.15 per ton.
For unloading, stocking and waste, ...	.30 per ton.
Making a total of, ...	$5.60 per ton.

or allowing 1¼ tons of coal per ton of pig-iron, $7 per ton. Actual contracts would probably materially reduce the above.

The present price of coke at the mines is above the average, but, basing an estimate on prevailing rates for Connellsville coke, we have

One ton of coke at Connellsville, ...	$2.00
Freight to Cleveland (estimated), ...	1.25
Dock charges at Cleveland (estimated), ...	.30
Freight to Duluth* (estimated), ...	1.25
Unloading and stocking (estimated), ...	.30
Allow for loss and waste, ...	.50
Making cost of one net ton of coke at Duluth, . . .	$5.60

or but 30 cents more per ton than is now paid at Chicago for coke brought by railroad.

It may be possible to get coke from other regions, *via* Cleveland, at less cost, and, in any actual contract, this would be considered.

If bituminous coal is to be used in an iron-works at Duluth for smelting purposes, it should be first coked; and the question of whether this will be done at Duluth will be determined by a comparison of the cost of a ton of coke made at the coal-mines and shipped by rail to the lake, and thence by vessel to Duluth, together with the cost of unloading and allowance for waste, with the cost of

* This freight rate is assumed because of the bulky character of coke as a vessel-load.

1.6 tons of coal at the mines, together with transportation-charges by rail and lake to Duluth, the cost of unloading, interest and depreciation on oven-plant, and cost of producing the coke, with the possibility of some reduction on account of utilization of waste heat or products of distillation.

The cost of coke made at Duluth is estimated as follows:

1.6 tons of coal at Cleveland at $1.40,	$2.24
Dock-charges at 25 cents per ton of coal,	.40
Lake freight to Duluth at 90 cents per ton of coal,	1.44
Unloading and stocking, at 20 cents per ton of coal,	.32
Allow for loss and waste,	.10
Cost of coking,	.60
Interest, depreciation and repairs on oven-plant,	.15
Making a total for the cost of one net ton of coke made at Duluth,	$5.25

An apparent saving over coke transported of 35 cents per ton.

This estimate is intended to cover the cost of employing a good coking coal which can be delivered at lake ports with low freight-charges.

The following estimate for charcoal is presented as representing the cost of coal made from one cord of wood:

Royalty or stumpage,	.30
Cutting wood,	.65
Handling to carbonization-works,	.80
Carbonization,	.50
Loading charcoal on cars,	.05
Interest and depreciation on plant,	.04
Hauling charcoal to furnace,	.45
	$2.79

Which, on the basis of 45 bushels per cord, will make the cost 6.2 cents per bushel of 20 pounds.

Under efficient management and with the excellent ores available, an allowance of two cords of wood (or, say, 90 bushels of charcoal) per ton of pig-iron should be sufficient, which would make the cost of fuel per gross ton of pig-iron $5.58, if charcoal was used in a plant well equipped and economically operated.

To produce a gross ton of pig-iron with the charcoal resulting from carbonizing two cords of hard wood, modern methods of carbonization must be used in addition to rich ores and good furnace-practice; but it is by no means impracticable.

By estimating 50 cents as excess freight from the Gogebic mines, and allowing 20 cents for transfer charges in Duluth, in either case we have the following as the cost at Duluth of one ton of—

Minnesota Bessemer ore (67½ per cent. iron),	$5.20 per gross ton.
Minnesota Red Lake ore (63 per cent. iron),	$4.20 per gross ton.
Gogebic Bessemer ores (60 per cent. iron),	$4.70 per gross ton.

Calculating on a mixture of Vermilion and Gogebic ores, the figures would show, if two-thirds Minnesota Bessemer were used with one-third Gogebic ores:

1.52 tons of ore at $4.87 per ton,	$7.40
Or, if two-thirds Red Lake and one-third Gogebic were used,	
1.6 tons of ore at $4.37 per ton,	$7.00

If all Minnesota Bessemer were used, which under the circumstances may not be desirable,

1.48 tons of ore at $5.20 per ton,	$7.70
If all Red Lake were used, at $4.20 per ton,	6.72

as the cost of ore per ton of iron.

These figures represent present prices at shipping-ports, which are considerably above current rates, and no account is taken of the future development of either region, which will probably bring a supply of ore still cheaper, nor is any allowance made for a possible reduction on all-the-year shipments as against the ore sent by lake when navigation is open. Therefore, the price assumed for ore per ton of iron will be rather excessive than otherwise.

Under the conditions existing, the following estimates for producing Bessemer pig-iron from the ores available at Duluth, and with the different fuels, are believed to be sufficiently liberal:

Cost of Producing One Gross Ton of Charcoal Pig-Iron.

1.52 gross tons of ore at $4.87 (two-thirds Minnesota Bessemer, one-third Gogebic),	$7.40
$\frac{3}{10}$ ton limestone at $1.50,	.45
90 bushels charcoal at $6\frac{3}{10}$ cents,	5.58
Labor,	1.45
Office and superintendence,	.40
Repairs,	.50
Supplies and incidentals,	.40
Interest and depreciation,	.80
	$16.98
If the product is sold to outside parties, the cost of sales and allowance for losses will be,	.60
	$17.58

The estimate of the cost of coke brought to Duluth is $5.10 per net ton, and an allowance for loss of handling is made of 50 cents per net ton, making the cost at furnace $5.60 per net ton.

The estimate for coke made at Duluth from coal brought there is $5.25 per net ton.

Taking a figure between these two estimates, and allowing interest on the stock necessarily carried through the winter, we may safely assume an average of $5.50 as the cost of a ton of coke at the furnace.

Cost of Producing One Gross Ton of Coke Pig-Iron.

1.52 gross tons of ore at $4.87 (two-thirds Minnesota Bessemer, one-third Gogebic),	$7.40
$\frac{4}{10}$ ton limestone at $1.50,	.60
One net ton of coke,	5.50
Labor,	1.35
Office and superintendence,	.40
Repairs,	.50
Supplies and incidentals,	.45
Interest and depreciation,	.90
	$17.10
Allow for sales and losses,	.60
	$17.70

Cost of Producing One Gross Ton of Anthracite Pig-Iron.

1.52 gross tons of ore at $4.87 (two-thirds Minnesota Bessemer, one-third Gogebic),	$7.40
$\frac{45}{100}$ tons of limestone at $1.50,	.68
1¼ tons of anthracite coal at $5.60,	7.00
Labor,	1.35
Office and superintendence,	.40
Repairs,	.50
Supplies and incidentals,	.45
Interest and depreciation,	.90
	$18.68
Allow for loss and sales,	.60
	$19.28

Without obtaining actual figures, we can compare the approximate partial costs of pig-iron manufacture at present rates for Chicago, Cleveland, and Duluth from the data above given; and we shall make the comparison on the use of coke, assuming that the same ores are employed at each place.

	Duluth.	Cleveland.	Chicago.
1 ton Minnesota ore at shipping port, .	$5.00	$5.00	$5 00
Freight, ...	.....	2.00	1.50
Dock charges or transfer, . . .	.20	.30	.30
$\frac{52}{100}$ tons Gogebic ore at shipping port, $4.00, ...	2.08	2.08	2.08
Freight, ...	.26	1.04	.78
Dock charges or transfer, . . .	.10	.15	.15
1 ton of coke at furnace, . . .	5.50	3.35	5 30
	$13.14	$13.92	$15.11

From this it would appear that, if the basis assumed is approximately correct, Duluth can obtain the same ores and fuel for smelting considerably cheaper than either Cleveland or Chicago, the saving being sufficient to allow liberally for increased cost of labor, due to advanced wages, comparatively small output or other contingencies. If all Minnesota ores are used the difference in favor of Duluth would appear still greater.

The great Northwest country tributary to Duluth is growing so rapidly that its requirements embrace nearly every species of iron and steel manufacture. To form a conception of the relative advance of the country naturally tributary to Duluth, or within as convenient access to it as to other lake ports, say, Chicago or Milwaukee, an examination of the railroad construction in 1886 is useful. Of the 8648 miles of railroad built in 1886, the following is credited to the states named:

	Miles.
Iowa, ...	379.68
Nebraska, ...	738.90
Wyoming, ...	117.90
Wisconsin, ...	369 36
Minnesota, ...	497.22
Dakota, ...	826.43
Montana, ...	18.00
A total of, ...	2,947.54

Or over 34 per cent. of the entire new mileage of the country.

The low rates of return freight, which make it possible to obtain a fuel-supply cheaply, would also to some extent encourage a competition from manufactures East and South, and the furnaces and mills of New York, Pennsylvania, Ohio, Michigan, Wisconsin and Illinois can obtain cheap freights on the products via vessels from Buffalo, Erie, Cleveland, Detroit, Chicago, Milwaukee and other lake ports, to Duluth. The freights on this class of goods are not as low as on coal,

but the outgoing and incoming tariffs would probably hold relative positions to bulk freights; and while they would practically prohibit an industry at Duluth shipping its products eastward, they would bring into competition with its manufactures others upon which an unusually low tariff would be charged. In addition to this, much of the territory looked upon as a legitimate market for Duluth's products is equally accessible to Chicago or Milwaukee, or St. Louis, by rail routes; and the Mississippi river will supply about as convenient means for handling the products of other localities as for distributing those of Duluth via St. Paul. But in a considerable territory Duluth will possess material advantages over other points, and it would seem that, under equal conditions, an industry at the head of Lake Superior could withstand any competition in products which were distributed from St. Paul and Minneapolis, both of which control large volumes of trade in iron and steel manufactures.

In addition, the section traversed by the Northern Pacific Railroad, and all territory north and west of a line drawn from St. Paul to Council Bluffs, Iowa, would naturally be most conveniently approached from Duluth until points more accessible from Denver or the Pacific slope were met.

And finally, from the data herein presented it would appear that the head of Lake Superior possesses advantages for the manufacture of iron in the abundance of excellent ores; the cheap coal, which is accessible by reason of back freights being less in amount than outward shipments (this will be continued, as the growth of grain and ore shipments will be greater than the demand for west-bound materials); the excellent and constantly improving market; the convenience of location; and the superior rail and water connections.

The data offered in this paper were collected in a number of professional visits to the Lake Superior district, four of which were made for the purpose of examining its ore-deposits, two to investigate the water-power of the St. Louis river, and lately one to consider the propriety of installing an iron- and steel-works at the head of Lake Superior.

Mr. W. F. Mattes, a member of the Institute, has also reported upon the last-mentioned subject; and his conclusions endorse the belief that the head of Lake Superior offers advantages which should place it among the prominent manufacturing centers. The subject is one which will naturally excite the interest of those who control our large iron and steel industries, by reason of the possibility of

RAINBLINE.—Plate I.

future competition from a region which, to one without any pecuniary interest in its development or business connection with its enterprises, seems destined to add still more than in the past to the wealth of our nation, and to advance in importance as its resources are developed and its advantages utilized.

NOTES.

In a report* of a "Geological Survey of Wisconsin, Iowa and Minnesota, made under instructions from the United States Treasury Department," and issued in 1852, Dr. J. G. Norwood gave much valuable information concerning Minnesota, among which is a description of a journey in 1847–8 up the St. Louis river, and into the Vermilion Lake district, supplying interesting topographical and geological data.

Col. Charles Whittlesey, who participated in this exploration, contributed in the same volume a chapter detailing investigations in Wisconsin, including the territory embraced in the Gogebic Range; he however reports finding magnetic iron-ore and no red hematite. After a lapse of forty years these records are full of interest to any who desire to investigate the subject to which the paper is devoted. A map of Lake Superior published in 1872 locates iron ores in what are now the Gogebic and Vermilion ranges.

THE INCLINE RAILWAY AT LOOKOUT MOUNTAIN.

BY W. H. ADAMS, M.E., NEW YORK CITY.

(Duluth Meeting, July, 1887.)

AMONG the engineering plants with new features and deserving details which are constantly being brought to the working stage in the Southern States by the generous expenditure of capital, none can exceed in interest, whether to the public or the profession, the incline-railway built from base to summit of Lookout Mountain, near Chattanooga, Tennessee.

Much geological and historical interest centers in this section of the middle mountain-country. Of its geology, the Transactions of the Institute contain many particulars. Of its history, the most

* Owen's Geological Survey of Wisconsin, Iowa and Minnesota.

recent and important part is fresh in our memories, and the region itself still abounds in evidences of the struggles for mastery and occupation of this strategic position during the late war.

Although it is situated on main lines of travel between the North and South, comparatively few of our people in search of health or recreation have availed themselves of the benefits and charms which latitude and altitude here offer. The expense and delays attending sight-seeing may heretofore have deterred many from visiting the spot, or the time may not have been opportune for calling attention to this section. Whatever the reason for past neglect in this respect, the time has come at last for a more thorough utilization of the scenic and sanitary resources of the region; and we have, as a result of combined talent and capital, a completed line for pleasure-travel which relegates to the past the wrangling driver of a licensed livery, or the long tiresome climb over dusty roads, to visit the glories of Lookout Mountain.

Volumes have been written, but no pen can describe the panorama of miles of undulating country traversed by the Tennessee river, and the scores of miles of lower mountain ranges which are spread out in every direction below the grand old mountain summit. Perhaps no single eyrie in the broad land can offer the charm of civilization and primitive grandeur so closely allied and so strongly blended as here from Lookout, the sentinel peak which guards the middle break in the southern Appalachian chain; and the opening of this railway system brings it within easy reach of all.

Six minutes of time will now transport the traveler from sordid earth to the ozone of the cloud-plane and unroll before him the magnificent view which has hitherto been obtainable only at the cost of a tedious ascent.

As compared with other inclines employed to overcome mountain heights, this work is a new departure; and the details here given by which many of the problems which vex the mining engineer also are cleverly solved will doubtless prove interesting to members of the Institute.

I am indebted to Major W. R. King, U. S. Engineers, for the details herewith presented, the project being his own, conceived during his residence in Chattanooga, and completed under his personal supervision. If Major King were one of the brotherhood of mining engineers, whose skill is so generally hidden from sight and sorely tried by the adverse circumstances of an underground existence, and had chosen an example to benefit his fellow-workers while

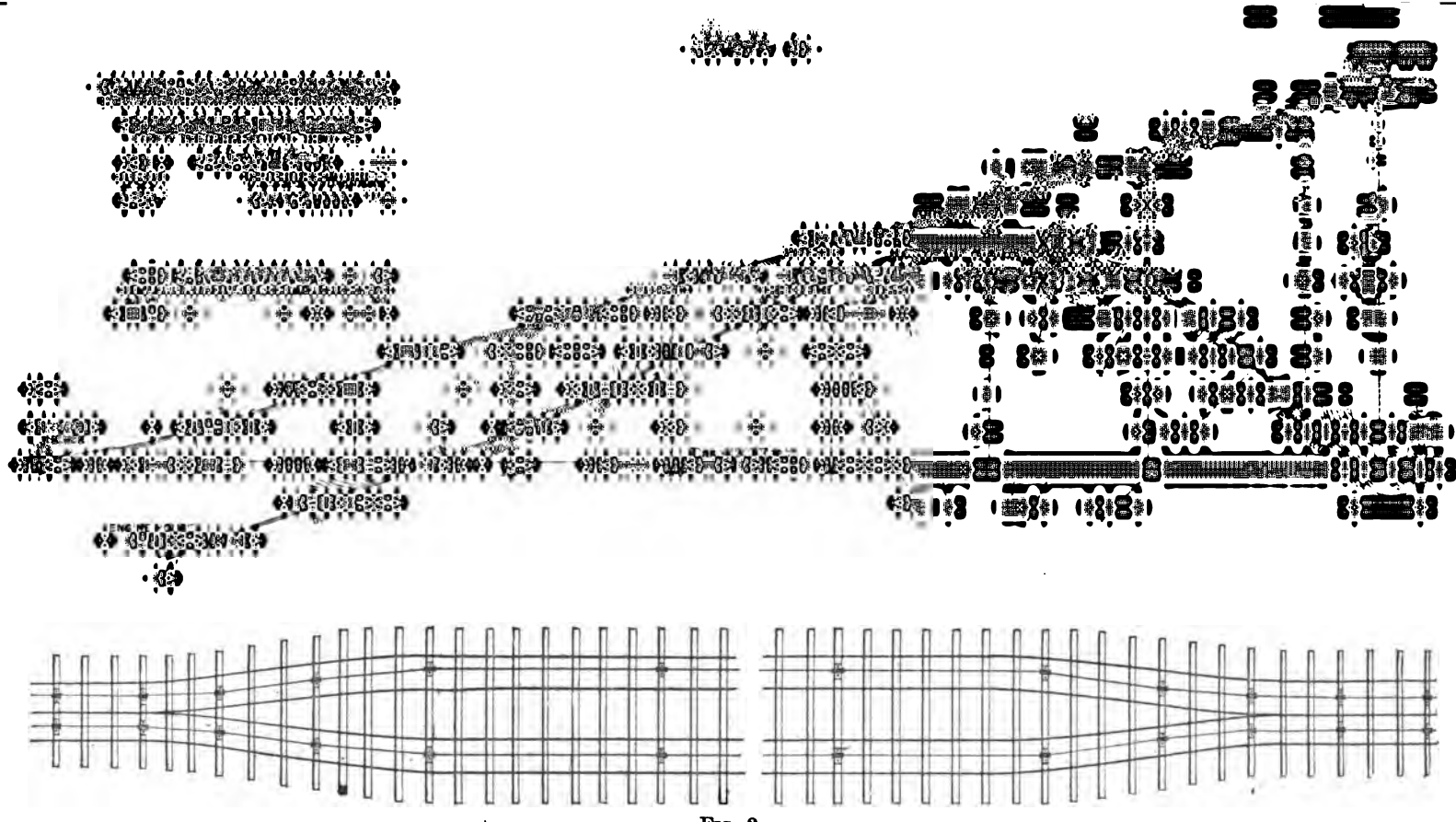


FIG. 2.

serving as the practical solution of a problem of great value to the public at large, he could not have offered a more genuine surprise than is met with in this railway, so quietly and expeditiously brought to a finish within the present year.

By reference to Fig. 1, which shows the railway in plan and profile, with cuts, fills, trestles, etc., it will be seen that the line is straight for 1250 feet, commencing at the base, curves to the left for 250 feet, thence runs in a tangent about 800 feet (which distance covers the switches or passing-points for the cars), thence curves to the right for 1400 feet, and finishes with a stretch of straight line for 600 feet to the base of bare rock which marks the crest of the mountain, 1500 feet above the river-level. The length of the track is 4360 feet, and the elevation attained is 1170 feet, or, say, one foot of rise to three and three-quarters of length. The problems required to be solved, the difficulties encountered in the nature of the surface, and the methods of overcoming all obstacles will be partially appreciated by reference to plans and photographs, but can be wholly understood by personal inspection only.

Cable lines in cities are the nearest approach to the methods here employed, but a close study of the special features which mark this new departure for surface-working will be of particular benefit to those engaged in mine-engineering.

Midway on the plan (Fig. 1, enlarged in Fig. 2*), the outer rails are shown as diverging, and the central rail opens out, or doubles, for a distance of 200 feet, forming two independent tracks or switches. This very unusual plan of a three-rail road, from end to end, was adopted for many reasons, prominent among which are:

1. No movable parts are used at any point in the track; therefore there is no risk of accidents from the failure of parts to work under the varying conditions of such service.

2. Solidity is assured by the extra width of the road-bed; therefore steadiness of running at high speed is practicable.

3. Sufficient space and play are given for the rapidly-moving cables to hold the center of the separated tracks at all times, the grooved guide-wheels, carefully spaced, holding the cars to the curves as evenly and truly as on the straight track.

The rails are of steel, 25 pounds to the yard, and laid on cedar ties nine feet long. No spikes are used, the rails being secured by heavy

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* The pulleys shown in Fig. 2 should be *between*, not *on*, the ties; and they are in reality much larger than they appear in this figure.

lag-screws five inches long, the heads of which are reinforced by wrought-iron washers 2 x 3 inches, which grip the base of the rail.

The cable consists of two sections. The upper one, 4360 feet long, and depended upon for the real work, is passed over an 8-foot sheave at the summit, and to its ends are attached the two cars, giving the well-known balanced hoist where power is applied at the upper end. In this case, owing to the necessity for applying the power to the lower end of the system, the ends are spliced below the cars to the two ends of a second section of cable, thus forming a continuous driving-rope. The center of the lower section passes over the grooved driving and tightening-pulleys, arranged as shown in Fig. 3.

A large surplus of strength is allowed for in the upper or working cable, the maximum load being five tons, while the breaking strain is fifty tons. This cable is 1¼ inches in diameter, and composed of six strands of nineteen wires each.

The driving-cable is one inch in diameter, with the same number of wires. The sheave at the summit is held in place by a wooden frame anchored to masonry piers which rest upon and are bolted to the original rock.

Fig. 3 shows a section and elevation of the engine-house and plant.

The engines, built by Wheeland, of Chattanooga, have two cylinders, 12 x 18 inches, connected to the main shaft, and a 20-inch steel pinion and two 80-inch gear-wheels actuate the two 80-inch double- and triple-grooved driving-sheaves.

The lead of the cable passes around the driving-sheaves, as shown in Fig. 3, and around a pair of smaller sheaves fixed in a sliding frame, to which is attached a counterpoise, serving to keep the cable uniformly taut, and allowing for changes of temperature or strains of any violent nature.

The engines are controlled absolutely by a system of levers extending into the tower of the engine-house, the engineer having an unobstructed view of the cars at nearly all points on the line.

The boilers carry 75 pounds working-pressure, and consume from 1600 to 2000 pounds of bituminous coal per day (the cars making from 18 to 24 trips), or about 90 pounds of coal per round trip.

The throttle, an ordinary 4-inch globe valve, is only opened from one- to two-thirds of a turn for the heaviest loads, while the full opening requires five complete turns.

With this small amount of steam the engines make about 160

revolutions per minute, the valves being set to cut off at two-thirds stroke. Late indicator-diagrams show that considerable improve-

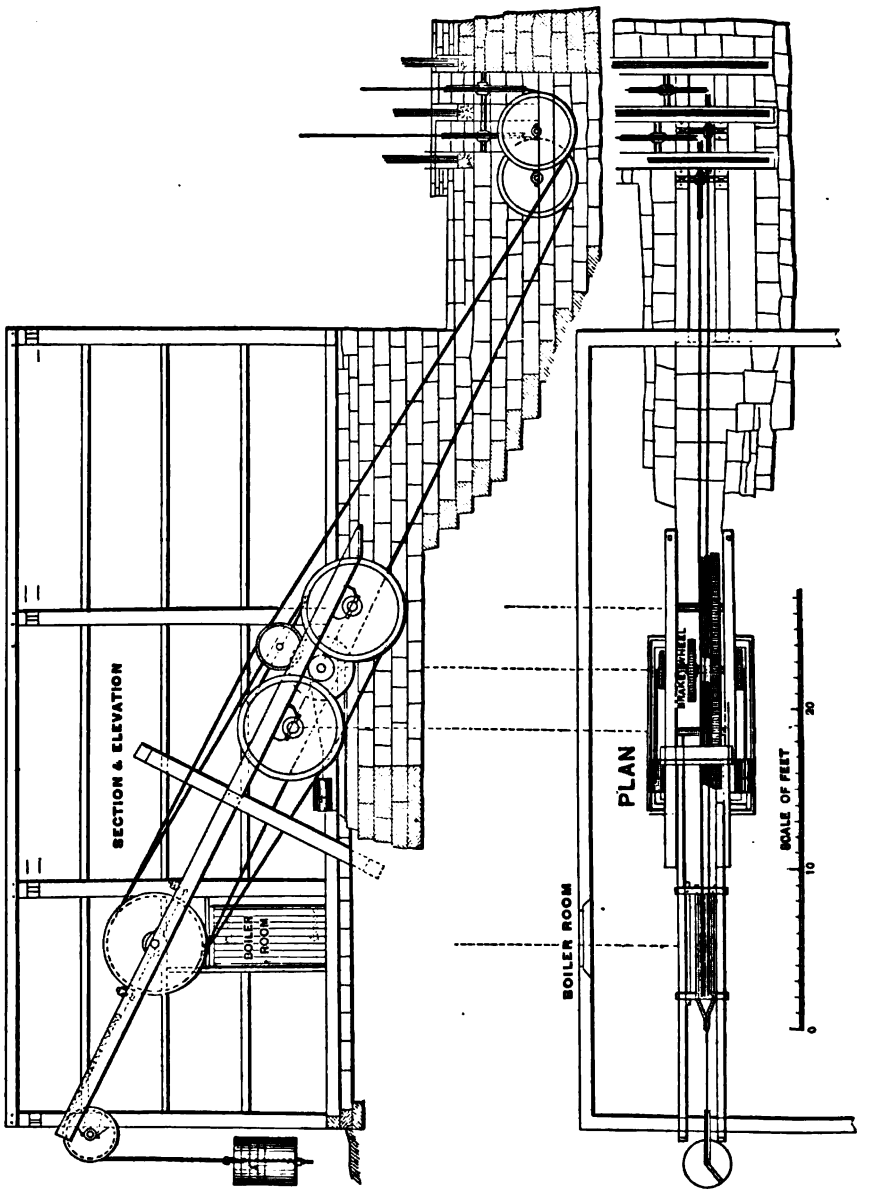


FIG. 3.

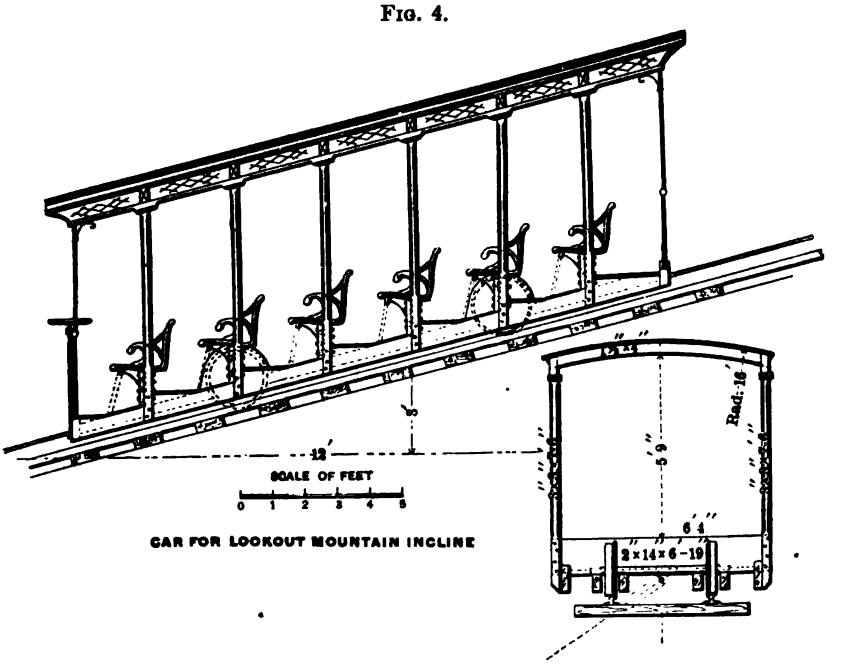
ment can yet be made in the setting of the valves; but these facts of present working will give a clear idea of the small amount of friction over the 9000 feet of wire rope. The expenditure of power

is surprisingly slight, when account is taken of the stiffness of cordage, weight of cable resting upon and moved rapidly over 150 grooved pulleys, passed at right angles through the power-plant, head- and tail-pulleys, etc.

The importance of telegraphic relations between the cars and engine-room was duly considered, and a very simple plan was devised by which either conductor can signal directly to the engineer from any part of the line whether the car is moving or standing still.

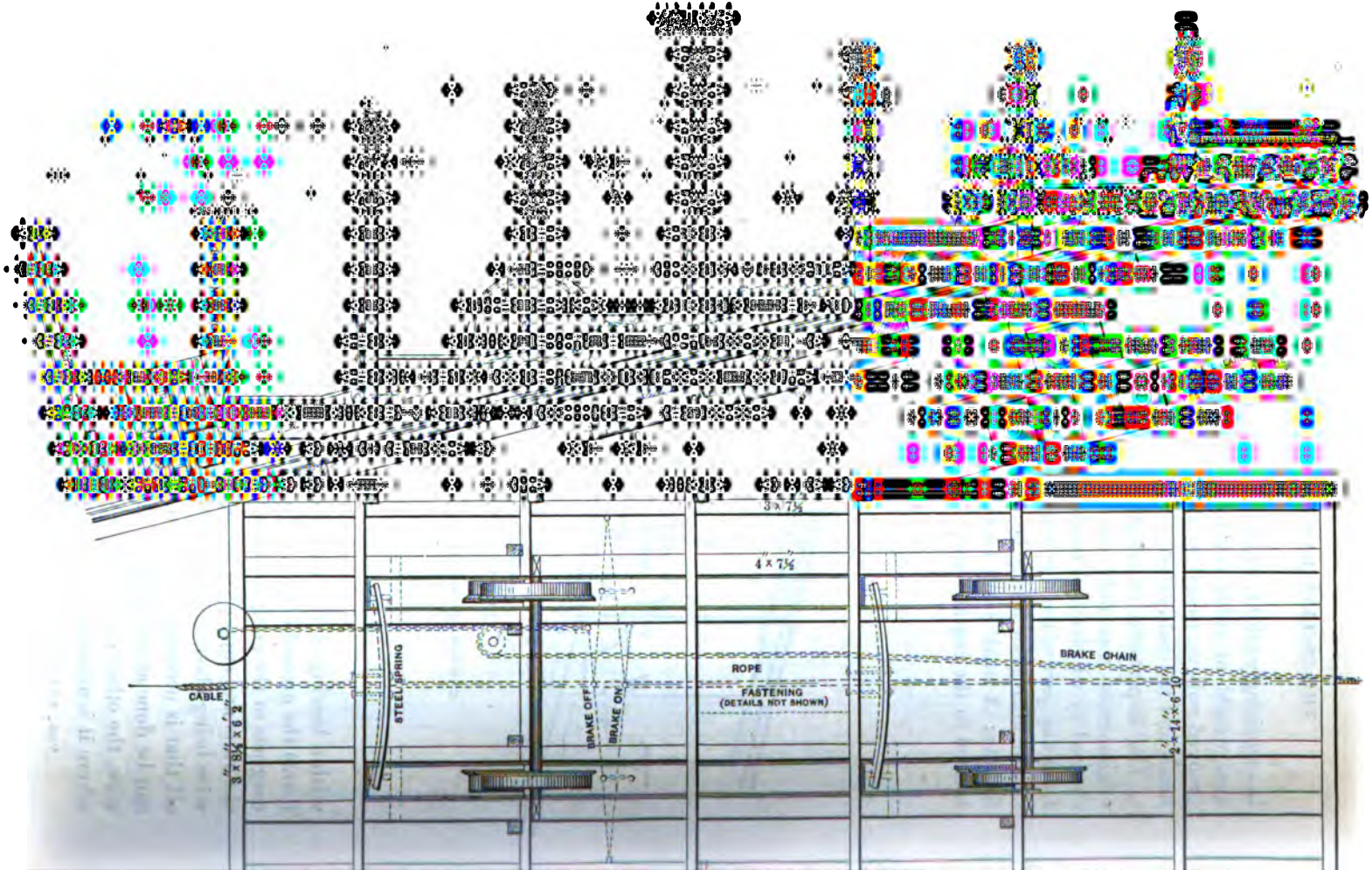
The arrangement may be readily described without drawings, as follows: A small 8-cell Leclanche battery and an electric gong are located in the engine-room. One pole of the battery is connected

FIG. 4.



CAR FOR LOOKOUT MOUNTAIN INCLINE

with a bearing of the main sheaves and thus with the cable, which forms the ground circuit. The other pole is connected through the magnets of the gong to an ordinary telegraph wire stretched between the rails and supported by insulators *that do not rise above it*, the wire being about six inches above the ties. To complete the circuit all that is necessary is to connect this wire with the cable, and this can be done at any point of the line by pressing a spring upon the wire, the other end of the spring being connected with the cable where it is attached to the car.


3 x 7½
4 x 7½
STEEL SPRING
BRAKE OFF
BRAKE ON
ROPE
FASTENING
(DETAILS NOT SHOWN)
BRAKE CHAIN
CABLE
3 x 8½ x 6' 2"
2 - 1¼" x 6' 10"

To guard against mistakes or possible failures to make the proper signal when in motion, it is understood *that any signal whatever*, when the engine is in motion, means "stop!" but a "start" requires a certain definite signal which can of course be easily and deliberately made, as the car must be standing still when such a signal is required. This last signal is such that it could hardly be counterfeited by accidental contact of the wires or by malicious persons. It should be noticed that, even if the cable were to be broken at any point, there will always be a connection one way or the other back to the engine-house, and if a car should jump the track the spring would come in contact with the insulated wire and thus give an instant signal to stop the engine.

Fig. 4 shows the construction of the cars, which were built as low as possible, and are provided with a special form of brake shown in Fig. 5. This brake acts in an entirely different manner from the common style of car-brake, the conductor being obliged to hold it off during the entire trip, or whenever the car is in motion. If, by accident or design, the hand-wheel is released, the brake is instantly forced under and against the front of each wheel, so as to lift the wheels entirely off the track, thus converting the car into a sled, of which the brakes are the runners.

The bottoms of the brakes now have soles of wrought-iron with short steel pins (24 in number) projecting slightly from the surface and sharpened like an engraver's tool. It is proposed to try a shoe with a A-shaped groove to take the head of the rail, the amount of friction being increased by the sharpness of the angle into which the head of the rail will be wedged by the weight of the car and load.

An automatic attachment causes the brake to set in the same manner in case the cable should part, even if the conductor should continue to hold on to the wheel; and still another and independent automatic grip is contemplated, to be actuated by a governor attached to the car-axle.

It should be added that the springs which actuate the brakes are not required to be bent more than six to eight minutes at a time, and will not be likely to take a "set," as is the case with springs on some forms of hoisting machinery which are kept under strain for months at a time.

Fig. 6 is a view on the upper portion of the incline, showing a car crossing one of the trestles. The oblique wheels carrying the cable are also seen. As this illustration is a Levytype from a photographic negative, taken on a bright day, the shadows of the two

cables are very distinct. They should not be mistaken for extra cables.

Fig. 7 is a view from the lower terminal, showing in perspective nearly the whole of the line. By comparison with Fig. 1, it gives a good notion of the work and of the topographical difficulties surmounted.

WIRE ROPE HAULAGE AND ITS APPLICATION TO MINING.

BY FRANK C. ROBERTS, C.E., PHILADELPHIA, PA.

(Duluth Meeting, July, 1887.)

PROGRESS in the facilities for handling mining products has been largely superinduced by the necessities of commercial economy rendered requisite in order to meet the demand of competition. So rapid has been the depreciation in the value of mineral products, primarily due to the disproportionate increase of output over consumption, that it may be justly claimed that mining, when considered as a commercial success, depends largely upon the ease and cheapness by which the products are brought to the surface of the ground. In all methods of performing this operation, wire rope enters as an important factor; and the object of the present sketch will be to explain, as fully as is consistent with the space allotted, the various adaptations of wire rope employed in placing coal and ores within the reach of our overground systems of transportation.

These adaptations will be considered in the following order:

I. Hoists.
II. Inclined planes.
  a. Engine planes.
  b. Gravity planes.
  c. Aerial planes.
III. Haulage.
  a. Tail-rope system.
  b. Counter-rope system.
  c. Endless-rope system.

I. HOISTS.

The term "hoist" is applied to the system whereby the mine-product is elevated from the mine-level to the surface through ver-

tical shafts. In our country this system finds its widest application in the anthracite coal-regions, where it is the prevailing method. The principle, briefly described, is as follows: A vertical shaft is sunk to the level which it is proposed to operate, and from this point the various workings penetrate. At the head of the shaft is placed an engine with a wire-rope drum attached. To this drum is fastened the hoisting-rope, which is led over a grooved wheel and attached to the carriage or cage upon which the cars rest while they are hoisted. The cage being at the foot of the shaft, loaded cars are run upon it and the signal is given to the engineer to start the hoisting-engine. In this manner the cars are hoisted to the surface and switched off to their destination.

Hoists may be divided into two classes, viz.: *single* and *double* hoists. The former have a single shaft and drum, with a non-reversing hoisting-engine, the cars being lowered by means of a friction-clutch attached to the rope-drum, and sufficiently powerful to hold the cage when loaded in any position. This system is exceedingly economical of fuel, the work of the engine being almost constant. The *double hoist* consists in a plant of two shafts or shaft-compartments and drums, with the hoisting-gear so regulated in operation that when an empty car is descending on one side a loaded car is ascending on the other. This arrangement requires, of course, a reversing engine; and although the plant is more expensive in instalment, it is to be recommended where large capacity is required.

Two types of hoisting-engines are constructed, viz., *first-* and *second*-motion engines. In the former, two engines with cranks at right angles to each other are coupled direct to the drum-shaft; in the latter, the engine carries a pinion, meshing with a spur-wheel keyed to the drum-shaft. The rope-drums in either case are proportioned in diameter to the size of rope employed, and in length to the depth of the shaft. In single shafts but one drum, of course, is necessary, but double-shaft hoisting-engines are provided with two rope-drums, each having its separate rope.

Fig. 1* illustrates what is known as the fusee or conical-spiral drum. Two of these drums are placed end to end on a common shaft with a strap-brake-seat between them. The wire-ropes are attached to the small ends of the drums and the engines wind one rope while unwinding the other. In this manner the leverage of

* Figs. 1 to 6, inclusive, are from the catalogue of the Dickson Manufacturing Co., of Scranton, Pa.

the rope performing the function of hoisting is diminished, while the descending rope has an increased leverage; and, as a result, the work done by the engine is more uniform and the gentle starting of the load is more easily accomplished.* These drums vary between 5 and 15 feet in diameter, and are of such lengths as may be suited to the depth of the mine.

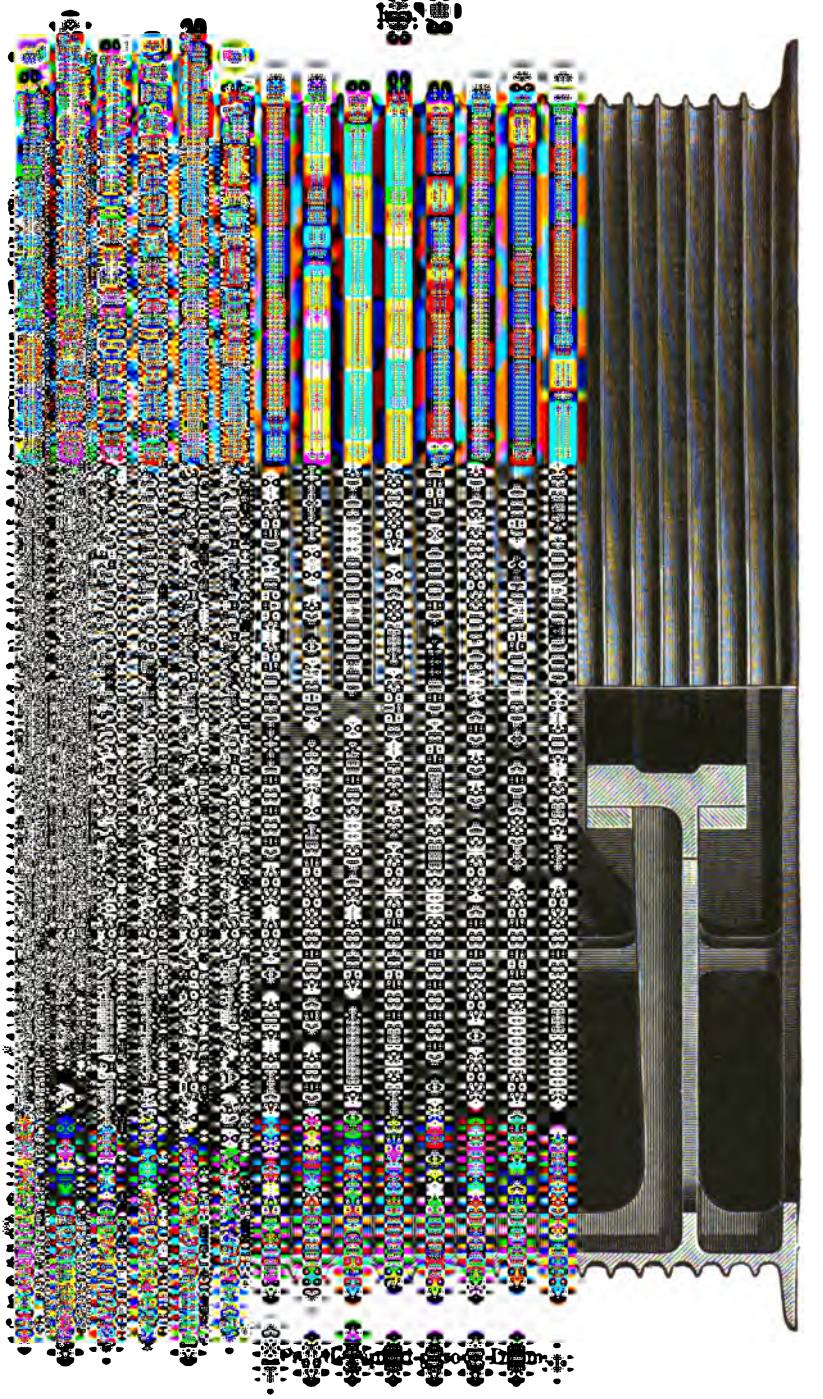
Fig. 2 represents the parallel-spiral-groove drum, which has a wide application, although lacking the advantage obtained in the fusee by the equalization of leverages. When applied to double hoists, two of these are keyed to one shaft, the end-flanges being bolted together. The ropes are attached as to the fusee, one winding while the other is unwinding.

In many mines or shafts, work may be progressing at different levels below the surface. Fig. 3 illustrates a design of hoisting-drum suitable to such requirements. It will be seen that, by means of a toothed wheel, the driving-power is transmitted to the drum, the arrangement being such that the gear meshes with a circular rack fixed in the end of the drum. When it is desired to adjust the length of the rope to a different level, the wheel is slipped out of the rack by means of a lever, and the drum is revolved until the proper length of rope is wound or unwound. The wheel is now returned to its former position, and the drum is ready for use.

In a vertical hoist, the wire rope, after passing off the drum, leads to what is known as the *head-sheave* (Fig. 4), which is so located that its outer circumference is directly over the center of the pit, and from which the rope leads vertically downwards into the mine and is attached to the *cage* upon which the mine-cars are hoisted to the surface. Fig. 5 represents a modern hoisting-cage, such as is used in the coal-regions. Upon two opposite sides of the pit are located the guide-bars, which are usually constructed of hard wood. In order to provide against accidents in case of the rope breaking, the cage is provided with safety-catches, the action of which is such that the instant the tension on the rope is released the quadrant-racks shown at the side of the cage imbed themselves in the guides and the cage is arrested in its descent. This movement is effected by a series of rubber springs set in telescopic distance-rings through which the draw-head-bolt passes, the whole being encased in the spring-pocket shown underneath the draw-beam. When the rope is in tension, the draw-head-bolt passes through and underneath the spring-pocket and compresses the springs; but upon the release of the tension in

* See tables, pp. 222 and 223.

[illegible] Drum.



head-bolt in toward the center
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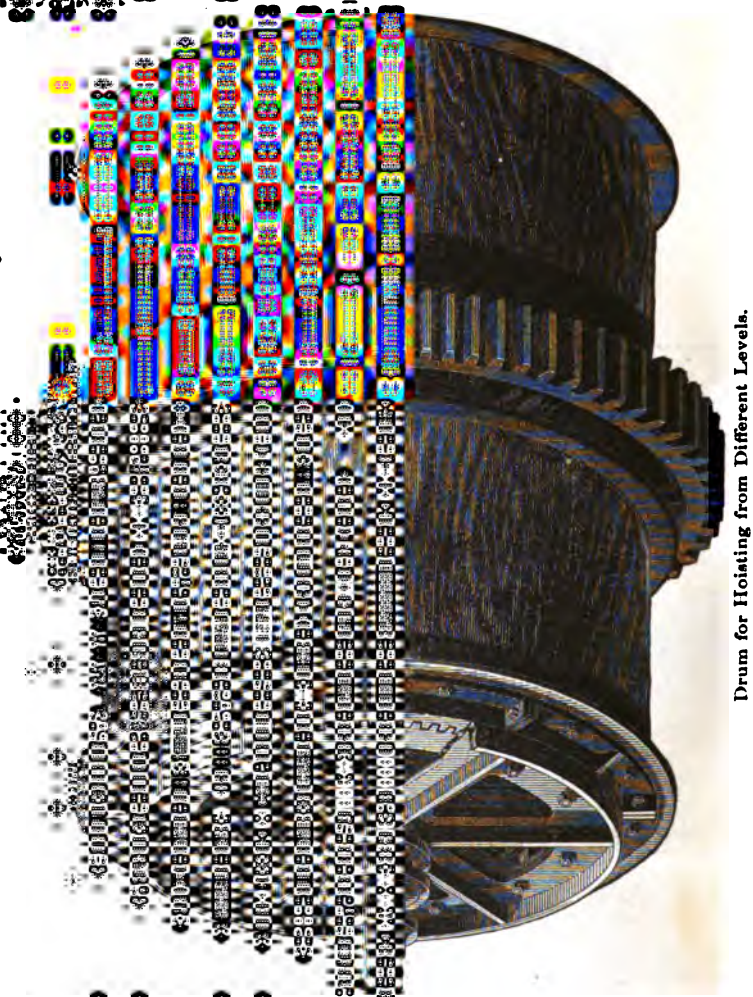

Drum for Hoisting from Different Levels.

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These consist of arms or prongs
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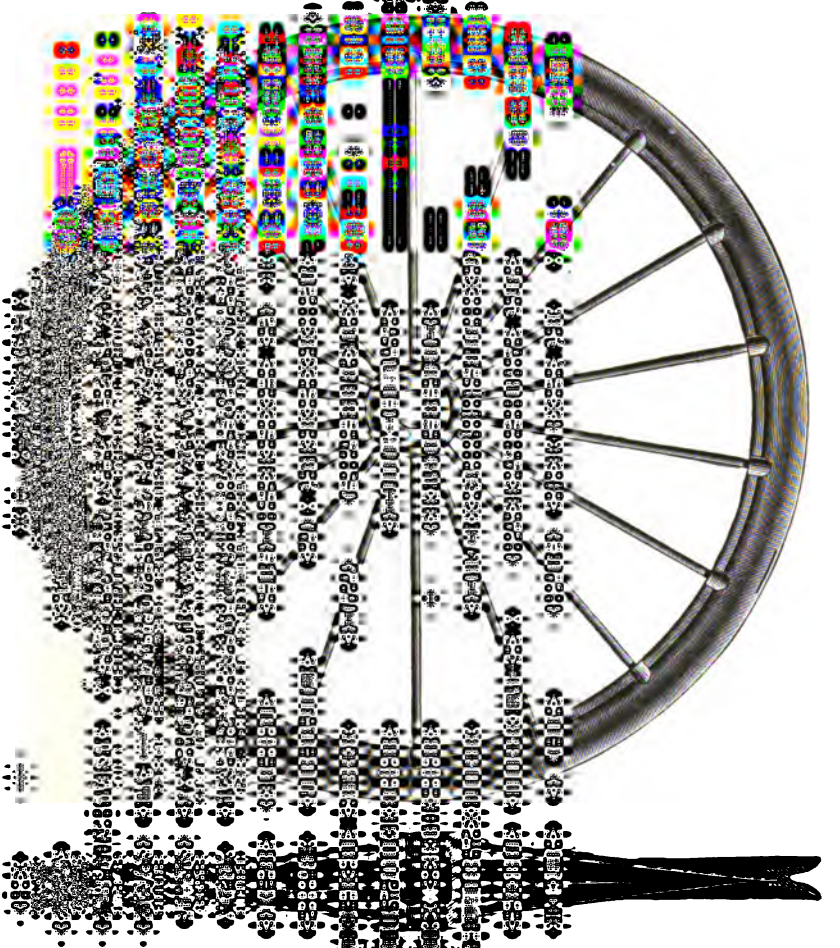

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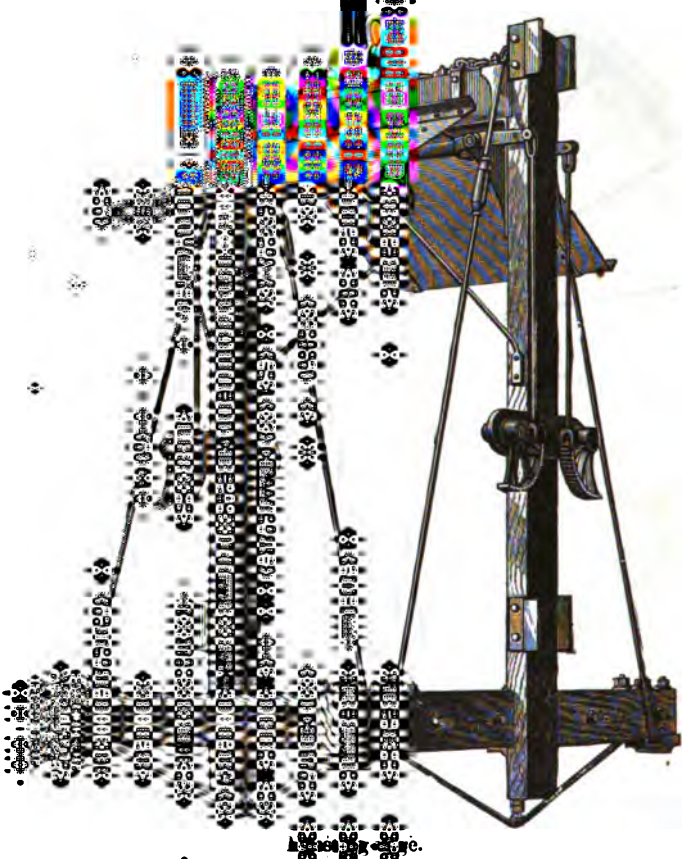

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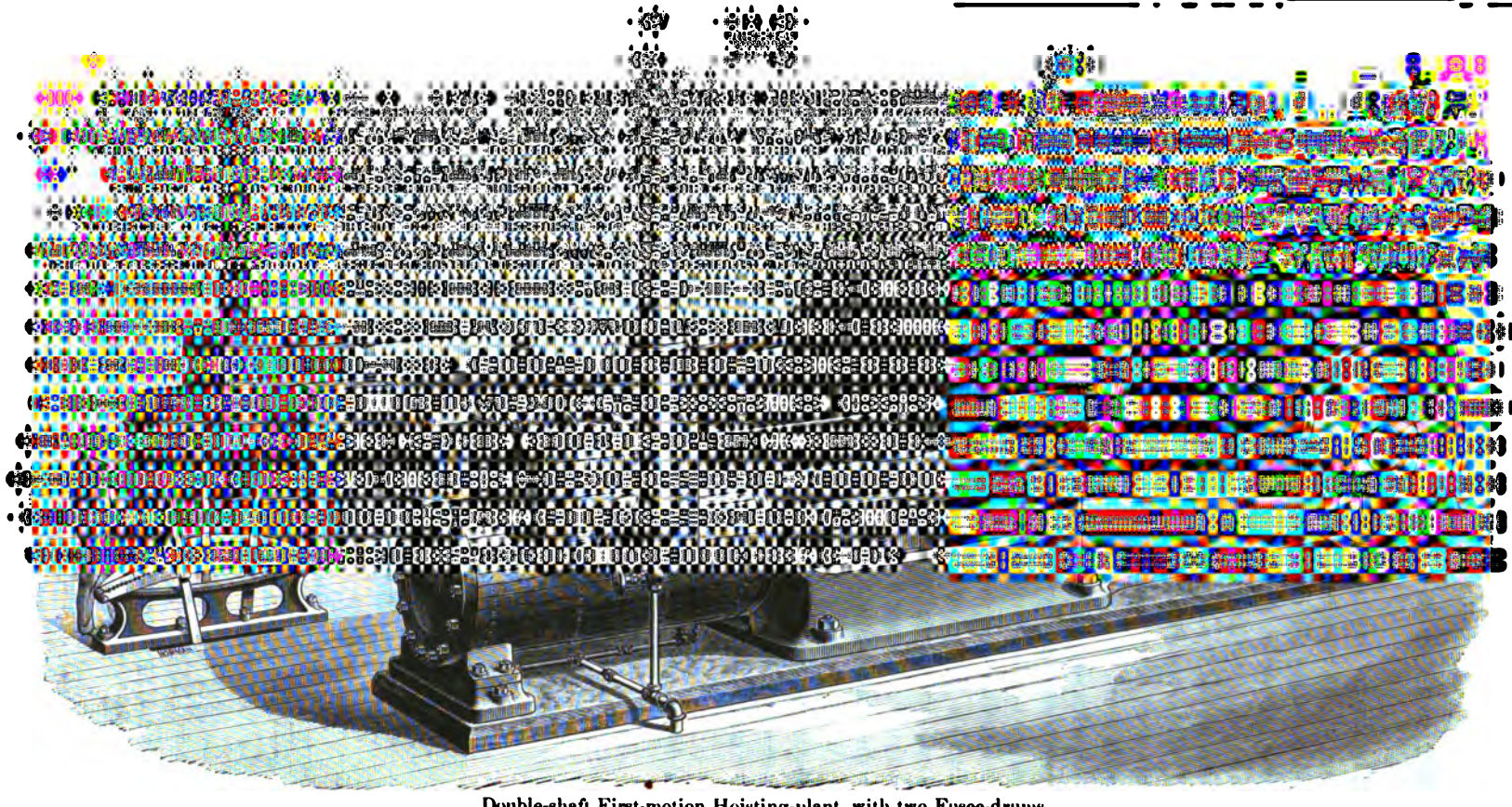
[illegible] dered, but also the friction of [illegible] eight of the rope itself. It is [illegible] ropean mine-operators to em- [illegible] *i.e.*, those having an increased [illegible] peculiarities of each applica- [illegible] of not less than five adopted.



Double-shaft First-motion Hoisting-plant, with two Fusee-drums.

The drums and head-sheaves must be properly proportioned to the size of the rope, or else the continual bending and unbending will soon destroy the elasticity of the wire. Nothing but the best Swedes iron or *reliable* steel rope should be employed. The danger to life and property is too great to permit the risk of non-uniformity in the material of the rope.

Wire rope for vertical hoists should be composed of six strands of nineteen wires each, wrapped about a hempen center. The latter adds much to the elasticity and life of the rope. Ropes of small diameter (up to $\frac{3}{4}$ inch), with six strands of seven wires each, are sometimes employed for hoisting purposes; but their use necessitates larger head-sheaves and drums and, consequently, increased cost.

Crucible steel ropes, when composed of good material, are more durable than iron ropes, and have entirely replaced the latter in some mining regions. It must be borne in mind, however, when making a change from iron to steel, that the aim should not be to reduce the diameter of the rope, but to increase its durability.

When starting the cage from the bottom of the shaft, the rope should be taut; the presence of slack causes a sudden and dangerous stress in the rope. The following table shows clearly the necessity for care in this direction:

Table of a Series of Tests Showing the Extra Stress upon a Hoisting-Rope due to a Few Inches of Slack Rope.

	Strain in pounds.
First Test.	
No. 1, Empty cage, lifted gently,	4,030
No. 2, " "	4,030
No. 1, With 2½ inches slack rope,	5,600
No. 2, " 2½ " " "	5,600
No. 1, " 6 " " "	8,950
No. 2, " 6 " " "	8,950
No. 1, " 12 " " "	12,300
No. 2, " 12 " " "	12,300
Second Test.	
Cage and four empty cars weighed by machine,	6,375
No. 1, Cage lifted gently,	6,725
No. 2, " " "	6,725
No. 1, With 3 inches slack rope,	11,200
No. 2, " 3 " " "	11,200
No. 1, " 6 " " "	12,250
No. 2, " 6 " " "	12,250
No. 1, " 12 " " "	15,675
No. 2, " 12 " " "	15,675

Third Test.

Cage and full cars weighed by machine,	11,300
No. 1, Cage lifted gently,	11,300
No. 2, " " "	11,525
No. 1, With 3 inches slack rope,	19,025
No. 2, " 3 " " "	19,025
No. 1, " 6 " " "	23,500
No. 2, " 6 " " "	25,750
No. 1, " 9 " " "	27,950
No. 2, " 9 " " "	25,750

The following table gives the necessary data to be used in selecting hoisting-ropes of nineteen wires to the strand. It is taken from the circular of the Trenton Iron Co.

Hoisting-Ropes.

Description.				Iron.				Steel.			
Trade No.	Diameter, in inches.	Circumference, in inches.	Estimated Weight per foot, in pounds.	Breaking Stress, in tons of 2,000 pounds.	Proper working load, in tons of 2,000 pounds.	Circumference of Hemp Rope of equal strength.	Min. size of drum or sheave, in ft.	Breaking Stress, in tons of 2,000 pounds.	Proper working load, in tons of 2,000 pounds.	Circumference of Hemp Rope of equal strength.	Min. size of drum or sheave, in ft.
1	2¼	7	7.75	74	15	15½	8	164.69	32.9	.....	9
2	2	6¼	6.11	65	13	14½	7	132.37	26.5	.....	8
3	1⅞	5½	5.09	54	11	13	6½	108.13	21.63	.....	7½
4	1¾	5	4.00	44	9	12	5	97.17	19.44	.....	6
5	1½	4⅝	3.55	39	8	11½	4⅝	86.38	17.3	16½	5½
5½	1⅜	4¼	2.90	33	6½	10½	4½	72.33	14.46	14	5¼
6	1¼	4	2.42	27	5½	9½	4	50.17	10.	12½	5
7	1⅛	3½	1.95	20	4	8	3½	38.	7.7	11	4½
8	1	3⅛	1.53	16	3	7	3	29.2	5.8	9	4
9	⅞	2¾	1.16	11½	2½	6	2¾	21.55	4.	8	3¾
10	¾	2⅜	0.85	8.64	1⅞	5	2½	14.99	3.	6½	3½
10½	⅝	2	0.60	5.13	1¼	4½	2	12.53	2.5	5½	3
10¾	⅝	1⅝	0.47	4.27	¾	4	1⅝	8.81	1.75	5¼	2¾
10⅞	½	1½	0.37	3.48	½	3½	1½	7.52	1.5	4⅝	2
10¼	½	1¼	0.26	2.50	¼	3	1				

II. Inclined Planes.

It is often found economical and convenient, when sinking a shaft, to incline the line of descent at an angle to the vertical, so that the material shall be hoisted on trucks running on track systems. Power is supplied by hoisting-engines, and for this reason these planes are denominated engine planes. Again, in many localities, especially in the bituminous coal region, the adits to the mine are at a considerable elevation above the overground system of transportation,

requiring the lowering of the mining-product from the mine to convenient points for shipment. This is usually secured by means of an inclined plane termed a gravity plane, and operated by the gravity of the descending load. In either type of inclined plane the line of descent may be curved or have variable grades, the only requisite being that the fall shall be in one direction and that the grade shall be sufficient to enable the loaded or empty car to descend by gravity.

Fig. 8 represents the arrangement of tracks most economical in construction for either system, while it also has the merit of a capacity equal to that of any other. The section C is termed the *parting*, its purpose being to enable the descending and ascending cars to pass each other. For this reason it is placed in the center of the inclined plane. The distinguishing feature of this road-bed is that above the parting we have three lines of rails, while below we have but two. This plan necessitates the use of an automatic switch at the lower point (A) of the parting (see Fig. 11, which is, however, reversed in position, as compared with Fig. 8). This simple arrangement consists in two iron-bound timbers pivoted at one end and moving over the rails. In the illustration the switch is arranged so that the loaded car going up will take the track S while the wheels of the descending car on M, in passing A, will shift the switch to the position shown in dotted lines. When the next loaded car ascends it passes into the parting on the track M. This arrangement of road-bed has been used with great success, and the general verdict of the mining community is that it answers, to all intents and purposes, the ends sought by more expensive devices. To avoid the necessity of the automatic switch at the lower end of the parting, the system illustrated in Fig. 9 has been devised. Its operation will be readily understood from the figure. The double-track incline, Fig. 10, is frequently employed, but the extra expense of broader road-bed renders it less common among mines of ordinary capacity. Where the output is very large, however, its advantages are evident.

An important detail in connection with all inclined planes is the arrangement of switches placed at the mine-end of the plane. Fig. 12 shows a simple and efficient device. The switch consists of three pivoted tongues, which govern, according to their position, the disposition of the cars. In the gravity system the loaded cars descend alternately by the tracks P and Q, and the empty cars always pass to N. In the engine-incline the loaded cars *ascend* alternately by the same tracks, the *loaded* cars here passing to N. As the switches

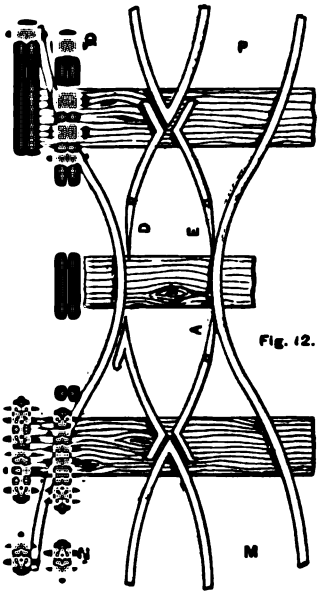


Fig. 12.

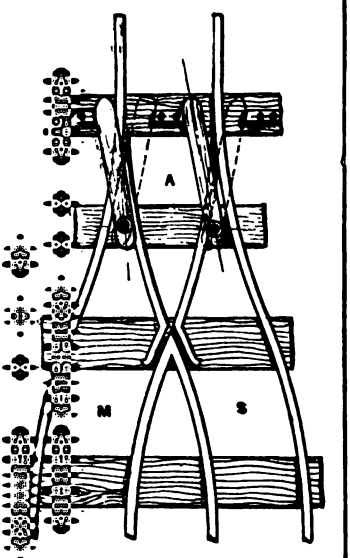


Fig. 11.

are placed, the empty cars, in the gravity-plane, will pass to N, and the loaded car advancing by M will pass to Q, while the empty car returning by P will open E and close D. A being closed, the empty car passes again to N, while the next loaded car advancing by M, descends the plane by P. Except that the tongue A has to be kept closed by a tender, the switch is automatic. The same is true of the operation of the switch at the head of an engine-plane, except that the loaded instead of the empty cars pass to N.

In all inclined planes it is necessary to employ some form of safety-device to provide against accidents arising from the breakage of the hoisting-rope. Much attention has been directed to these safety-devices, but in all designs the operation is dependent upon one of two principles—either to stop the cars or to guide them off the incline. The chief objection to these devices is that they are all controlled by a man stationed at the head of the plane. It is to be regretted that there exists no simple and inexpensive method whereby the safety arrangements are rendered automatic, as in the vertical hoists.

Fig. 13 represents a simple device for stopping runaway cars. Two heavy iron-bound timbers, A, are pivoted near their centers at the side of the outer rails, as shown. The arrangement of counterweights is such that when it becomes necessary to bring the timbers into action the pin P, forming the connection between the two timbers, is withdrawn by pulling the wire S, connecting with the head of the plane. As a result, the release of the counterweights W W throws the timbers over the rails, as indicated by the dotted lines. A number of these devices are placed along the incline, all being connected to the single wire leading to the head of the plane. This method is somewhat inferior to the one illustrated in Fig. 14, owing to the uncertainty of stopping the cars when descending at a great velocity, their tendency being to jump the timbers and continue in their course. The device shown in Fig. 14 consists in side-switching the car. This is performed by the tongues D, E, F and G, actuated by a system of bell-cranks and weights, as shown. The switch is held open by the pin P, and when this is withdrawn by means of the wire S, the switches are closed by the action of the counterweights W. This device, although resulting in the partial destruction of the car, renders the danger to life and valuable property considerably less than the preceding arrangement. What is known as the *dead-fall* is employed in some mines. A heavy timber, guided in an up-

Fig. 13.

right frame, is placed over the track, and, when released, falls over the rails.

In all applications of wire rope except vertical hoists, it is necessary to support the rope at frequent intervals by rollers, otherwise the rope would drag upon the ground and increase the power absorbed by friction, as well as cause the rope to wear out rapidly. A cheap and efficient roller (Figs. 15 and 16) is constructed of gum-wood, of

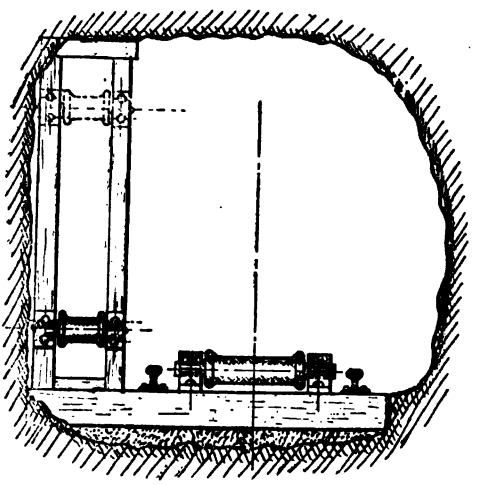


FIG. 15.

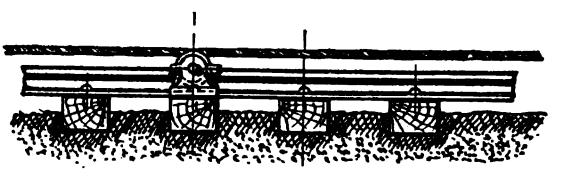


FIG. 16.

as large a diameter as is consistent with the location, and from 12 to 24 inches in length. The axles are 1 inch or 1¼ inch round iron, running in simple cast-iron bearings. The much-discussed question of comparative detrimental effect upon wire rope of wooden and iron rollers is doubtless to be answered in favor of the former; but whatever the advantage may be, it is so slight that it is hardly worth consideration, since, in a properly-constructed plane, the rope simply rests upon the rollers.

Curves.—The difficulty of passing curves is solved by two efficient

one device, whereby
ries of wheels about
The car, in turn-
but, after passing,

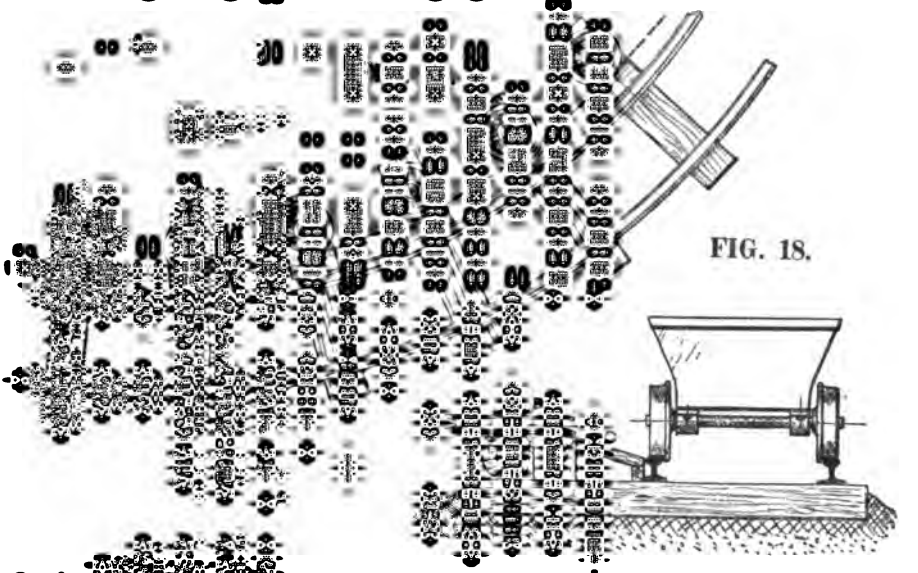


FIG. 18.

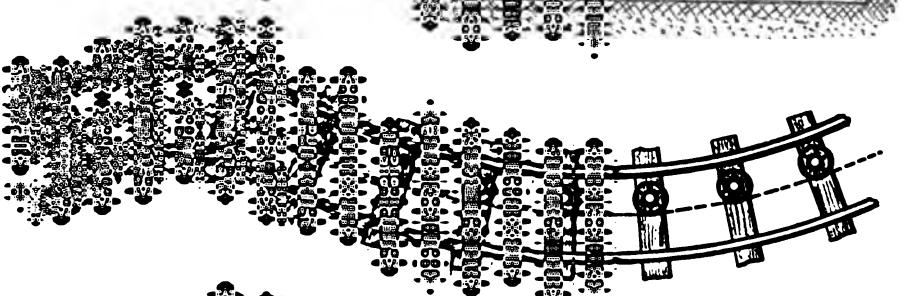


FIG. 19.

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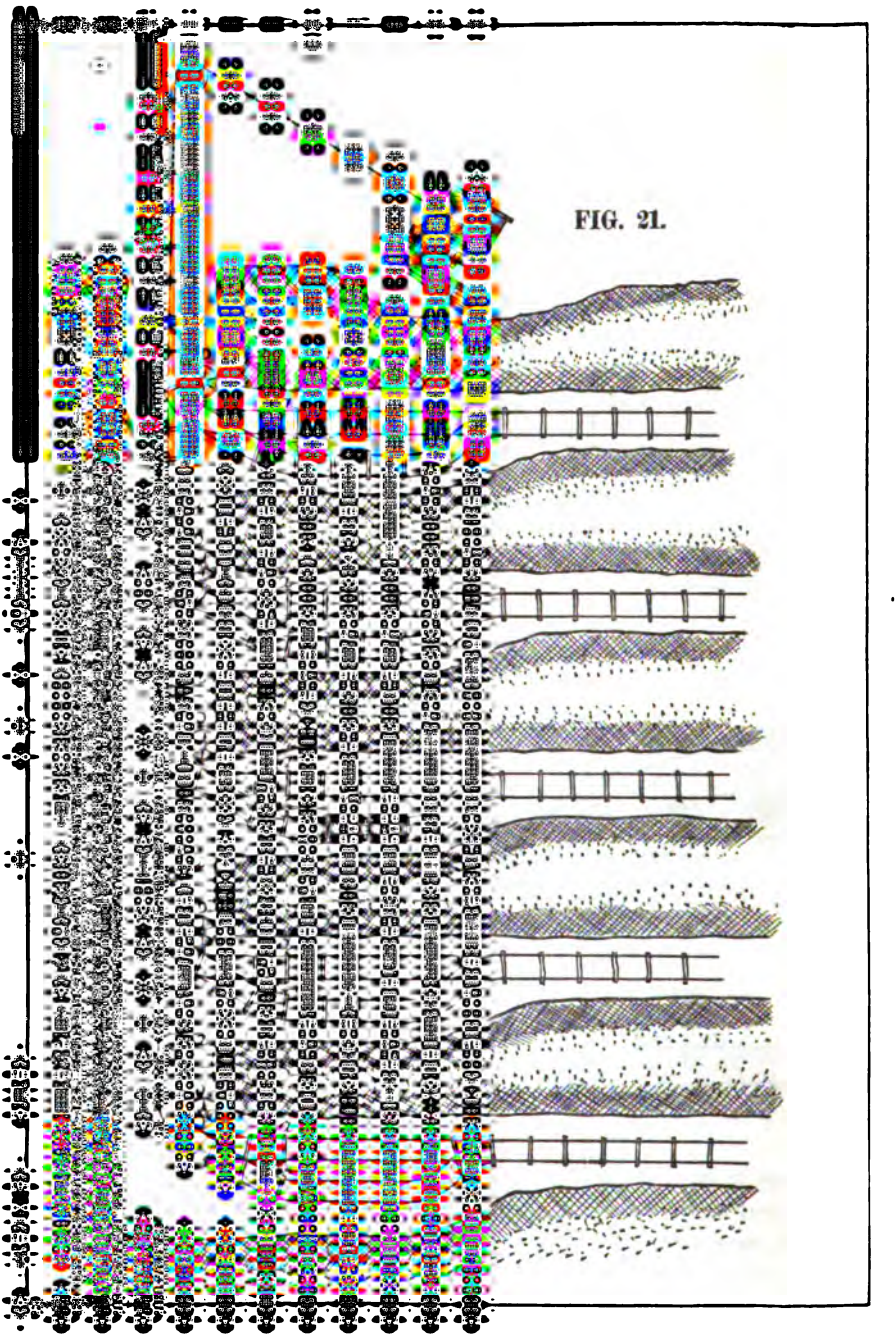

FIG. 21.

...oves of the wheels. ...er of the track, and ...en rounded. Either ...e, the determination

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FIG. 22.

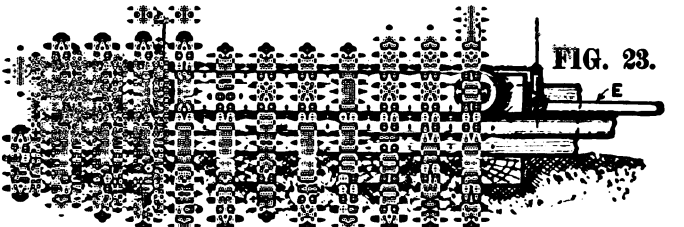


FIG. 23.

...ass, especially when ...are naturally divided ...oisting and lowering ...upon a single track ...provide for hoisting ...y necessitate three or

...y a non-reversing en- ...slope may be oper- ...t one-half the time. ...descend by gravity, ...ime and being con- ...it is sometimes con-

venient to locate the engine at the foot of the plane, the hoisting-rope being supported along the side of the slope by rollers (Figs. 15 and 16) and led to the head of the plane, where it passes around a wheel of large diameter and, returning down the track, is attached to the car.

The single-slope system is peculiarly applicable where there are a number of side-entries located at different levels below the surface. In this case the train of descending cars may be stopped at the entry into which it is desired to switch some of the trucks, the train in the meantime being prevented from premature starting by a lock consisting of a timber pivoted and thrown over the rails. When the disconnection has been made, the signal is given and the train descends to a lower level, where the operation may be repeated. In hoisting to the surface, the cars in the various side-entries are run to the main entry and, in a manner similar to that above described, switched to the main slope and connected to the rope.

An improved modification of the single plane is illustrated in Fig. 21. Here, for the accommodation of the empty or loaded cars, each side-entry has its individual parting or siding placed near the main entry, and in this way the plane is practically divided into as many separate planes as there are entries. Loaded cars are taken from each entry in succession, and the empties returned in the same order, the position of the end of the rope being indicated to the engineer by marks upon the rope, or better, by an indicator attached to the drum. Proper switches, etc., are located at the desired points, and the rope is guided into the entry by drums D, placed in the angles as shown. These drums have a face of two and a diameter of five feet. Guide rollers, *a*, are placed at suitable points.

Double Engine-Planes.—In double engine-planes, engines similar to those employed in double hoists are used, requiring, of course, reversing-gear and an efficient brake. The operation of these planes differs from the preceding only in the detailed arrangement of switches, sidings, etc. These devices may be readily understood by an examination of the various track-systems and other expedients already described.

II. *b.* GRAVITY-PLANES.—The valley of the Monongahela is celebrated for its gravity-inclines. Here the mine-adits are located at a considerable elevation above the railroads, the incline from the pit-mouth to the railroad being of such a degree that loaded mine-cars, descending by gravity, raise a corresponding number of empty

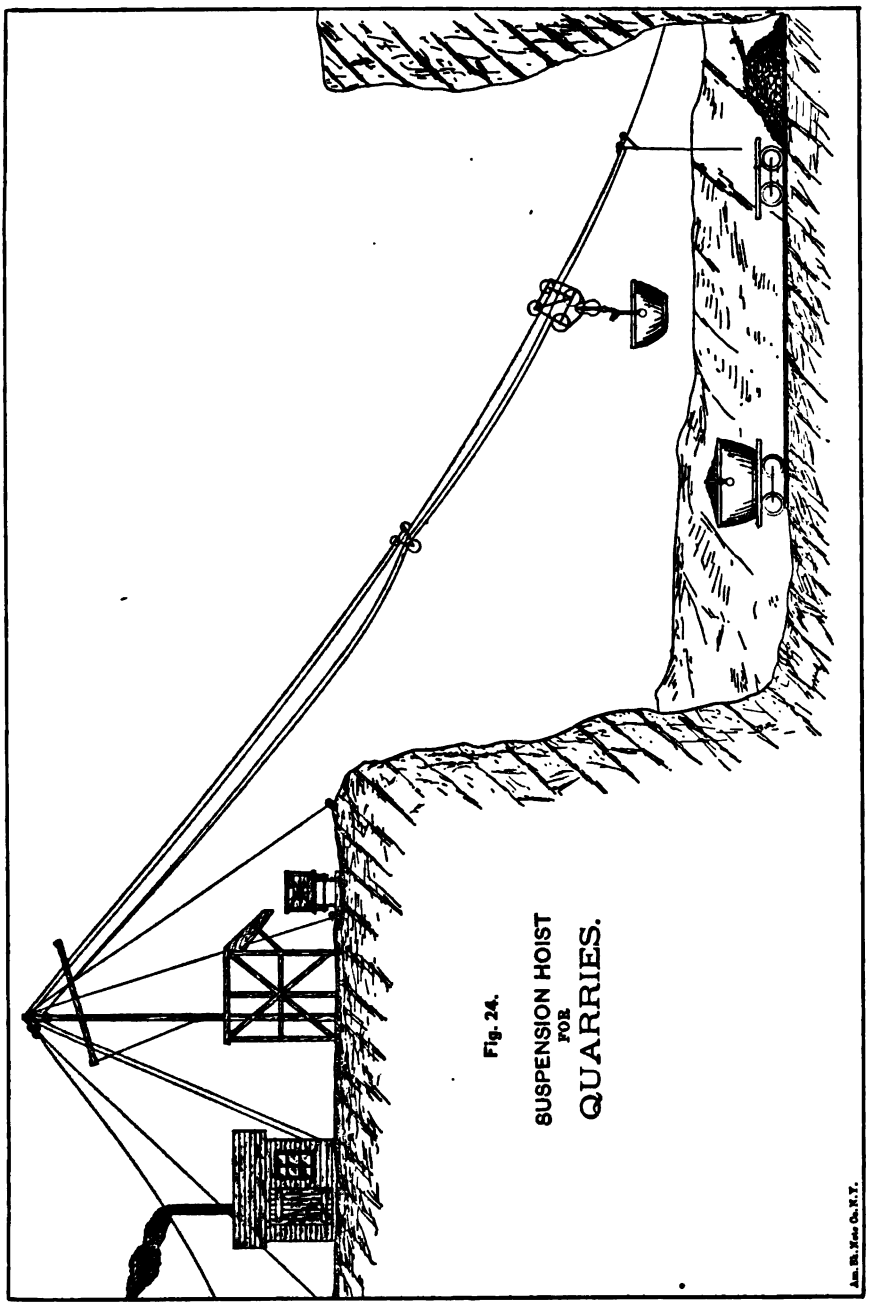


Fig. 24.

SUSPENSION HOIST FOR QUARRIES.

cars. To this application of natural force the name "gravity-plane" has been given. Perhaps to the ordinary observer the profile of the gravity-incline is a matter of small moment, and yet this factor is important in securing a satisfactory operation. The plane should be slightly concave. Theoretically the curve is a cycloid, the peculiar property of this curve being that a body falling along its lines will reach the lowest point of the plane sooner than by any other course, straight or curved. It can also be demonstrated that when the cycloid is employed, the weight of the two ropes is equalized at every point, so that the resistance from this cause is constant. On a straight incline, a variable braking-force is required; but on a cycloidal incline the brake may be applied with equal force during the whole time of descent, and a uniform rate of motion insured. It would, of course, be a difficult matter to build the profile of a plane so as to conform to cycloidal lines; but, for the reasons mentioned, it is advisable to approach as nearly thereto as possible.

The various track-systems common to both engine- and gravity-planes have been described above; but while the three- and four-rail systems are similar in principles of operation, the single-track system (Fig. 7) as applied to gravity-planes is modified to such a degree as to require special consideration. This modification consists in employing two tracks of different gauge, the narrower being placed within and below the wider (Figs. 22 and 23). The wider track is for the pit-cars, while the narrower serves as a roadway for what is termed the balance-car, C. The latter is so proportioned in weight that it is sufficient to raise an empty car and light enough to be elevated by the descending loaded car. To the balance-car is attached a safety-catch E, as shown in Fig. 23. When the rope breaks, or through any means becomes slackened, the lever E drops, and, coming in contact with the ties of the road-bed, prevents rapid and dangerous descent.

Gravity-planes are operated by either one or two wire ropes. Where a single rope is employed, it receives three or four turns around the head-drum, in order to prevent slipping, and the ends are attached to the empty and loaded cars respectively. (In the single-track system one end is, of course, attached to the counter-balance car.) This arrangement is very crude and not to be recommended under any circumstances. There is an unavoidable sideward movement in the rope, which not only causes great inconvenience but is also the source of rapid wear to the rope. Where the employment of a single rope is necessary, it will be found more

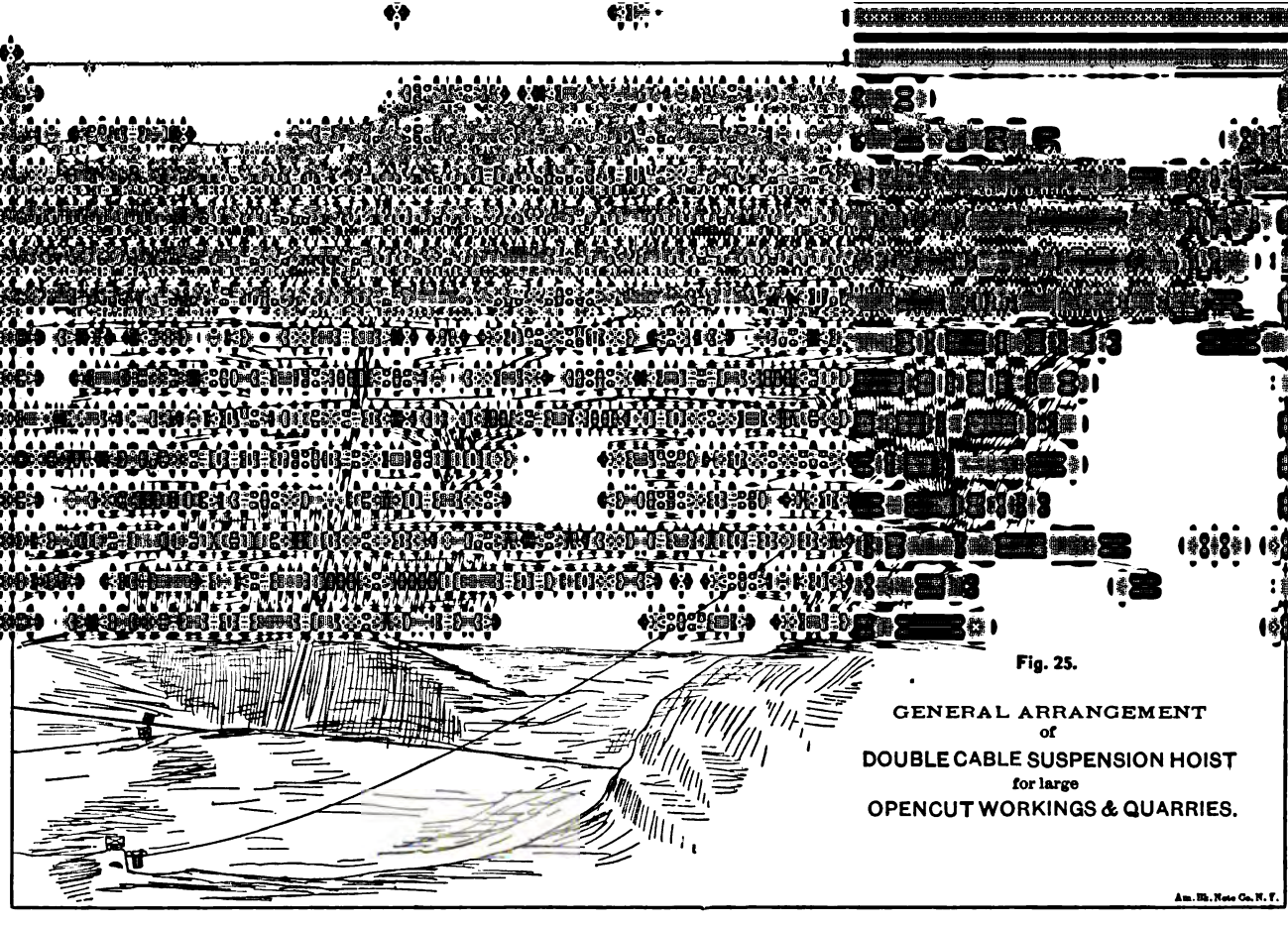


Fig. 25.

GENERAL ARRANGEMENT
of
DOUBLE CABLE SUSPENSION HOIST
for large
OPENCUT WORKINGS & QUARRIES.

Am. Bk. Note Co. N. Y.

advantageous to employ a standard wire-rope grip-sheave. The latter consists in an iron sheave, having its rope-seat composed of a number of hinged segments. These are arranged in such a manner that the pressure of the rope causes the upper part of the segments to close and grip the rope. The preferable method, however, is to employ two ropes. These may be operated in various ways. In the first instance, we may employ one drum, the rope winding and unwinding from opposite ends, or again, we may have two smaller drums keyed to one shaft, each having its own rope. Both these expedients require that the two ropes shall pass off at opposite sides of the drum, *i.e.*, one at the top and the other at the bottom. Where economy of space is requisite at the heading, this becomes a very inconvenient arrangement. In order to avoid this difficulty, two drums mounted on separate shafts and connected by gear-wheels are used. In this manner it becomes possible to lead both ropes from the under or top side of the drums as may be desired. The most complete device is to employ two fusee-drums, placed end to end on one shaft with the brake-seat between them. These may be located at some distance back from the head of the incline. Two head-sheaves are here employed, over which the rope passes and continues down the incline.

In all gravity-planes it is necessary to employ a brake-apparatus attached to the rope drum in order to check* the speed of the descending cars. Any approved brake may be used for this purpose, care being taken to apply a device that may be operated speedily and with little exertion. The speed of the drums must be entirely under control.

The same precautions should be taken in proportioning a wire rope, to be used on inclined planes, as have been specified under hoists. Wire rope of six strands of seven wires each, laid about a hempen center, is the most serviceable on inclined planes. The wire in this grade of rope is coarser than in the 19-wired rope, and is consequently better able to withstand the rough usage and surface-wear encountered in this application.

Fig. 39 illustrates the most approved design of swivel-chain-connection between the wire rope and cars on inclined planes.

The stress in the rope when applied to inclined planes is dependent on the inclination to the horizontal. The following table will be of assistance in determining the stress; but it must be borne in

* From this term is derived the expression "checker house," applied to the shed covering the head-drums, in which the operation of "checking" is done.

Arrangement of Terminal for Double Cable Suspension Hoist.

mind that, while the table is based upon an allowance of 40 pounds per ton for rolling friction, there will be an additional stress, due to the weight of the rope, proportional to the length of the plane.

Stress in Hoisting Ropes on Inclined Planes of Various Degrees.

Rise per 100 ft. horizontal.	Angle of Inclination.	Stress in pounds per ton of 2,000 pounds.	Rise per 100 ft. horizontal.	Angle of Inclination.	Stress in pounds per ton of 2,000 pounds.
Ft.			Ft.		
5	2° 52′	140	105	46° 24′	1484
10	5° 43′	240	110	47° 44′	1516
15	8° 32′	336	115	49° 00′	1535
20	11° 10′	432	120	50° 12′	1573
25	14° 03′	527	125	51° 21′	1597
30	16° 42′	613	130	52° 26′	1620
35	19° 18′	700	135	53° 29′	1642
40	21° 49′	782	140	54° 28′	1663
45	24° 14′	860	145	55° 25′	1682
50	26° 34′	933	150	56° 19′	1699
55	28° 49′	1003	155	57° 11′	1715
60	30° 58′	1067	160	58° 00′	1730
65	33° 02′	1128	165	58° 47′	1744
70	35° 00′	1185	170	59° 33′	1758
75	36° 53′	1238	175	60° 16′	1771
80	38° 40′	1287	180	60° 57′	1782
85	40° 22′	1332	185	61° 37′	1794
90	42° 00′	1375	190	62° 15′	1804
95	43° 32′	1415	195	62° 52′	1813
100	45° 00′	1450	200	63° 27′	1822

I append also a table (Trenton Iron Co.), giving the various data necessary to select a wire rope to meet specified requirements.

Description.				Iron.			Steel.		
Trade No.	Diameter, in inches.	Circumference, in inches.	Estimated Weight per foot, in pounds.	Breaking Stress, in tons of 2,000 pounds.	Proper working load, in tons of 2,000 pounds.	Circumference of Hemp Rope of equal strength.	Breaking Stress, in tons of 2,000 pounds.	Proper working load.	Circumference of Hemp Rope of equal strength.
11	1 1/2	4 3/4	3.37	36	9	10 3/4	88.38	22.	16 1/2
12	1 3/8	4 1/4	2.77	30	7 1/2	10	67.2	16.8	15 1/2
13	1 1/4	4	2.28	25	6 1/4	9 1/4	60.67	15.2	15
14	1 1/8	3 1/2	1.82	20	5	8	39.84	10.	11
15	1	3 1/8	1.50	16	4	7	31.82	8.	9 1/2
16	7/8	2 3/4	1.12	12.3	3	6 1/2	24.7	6.2	8 1/2
17	3/4	2 3/8	0.88	8.8	2 1/4	5 1/4	18.48	4.6	7 1/4
18	11/16	2 1/8	0.70	7.6	2	5	16.32	4.	7 1/4
19	5/8	2	0.57	5.8	1 1/2	4 3/4	12.44	3.1	6
20	9/16	1 3/4	0.41	4.1	1	4	9.33	2.3	5 1/4
21	1/2	1 1/2	0.31	2.83	3/4	3 1/4	6.89	1.7	4 1/2
22	7/16	1 3/8	0.23	2.13	1/2	2 3/4	5.23	1.3	3 1/2
23	3/8	1 1/8	0.19	1.65	...	2 1/2	3.93	1.	3 1/8
24	5/16	1	0.16	1.38	...	2 1/4	3.25	.81	3
25	1/4	7/8	0.125	1.03	...	2	2.96	.75	2 3/4

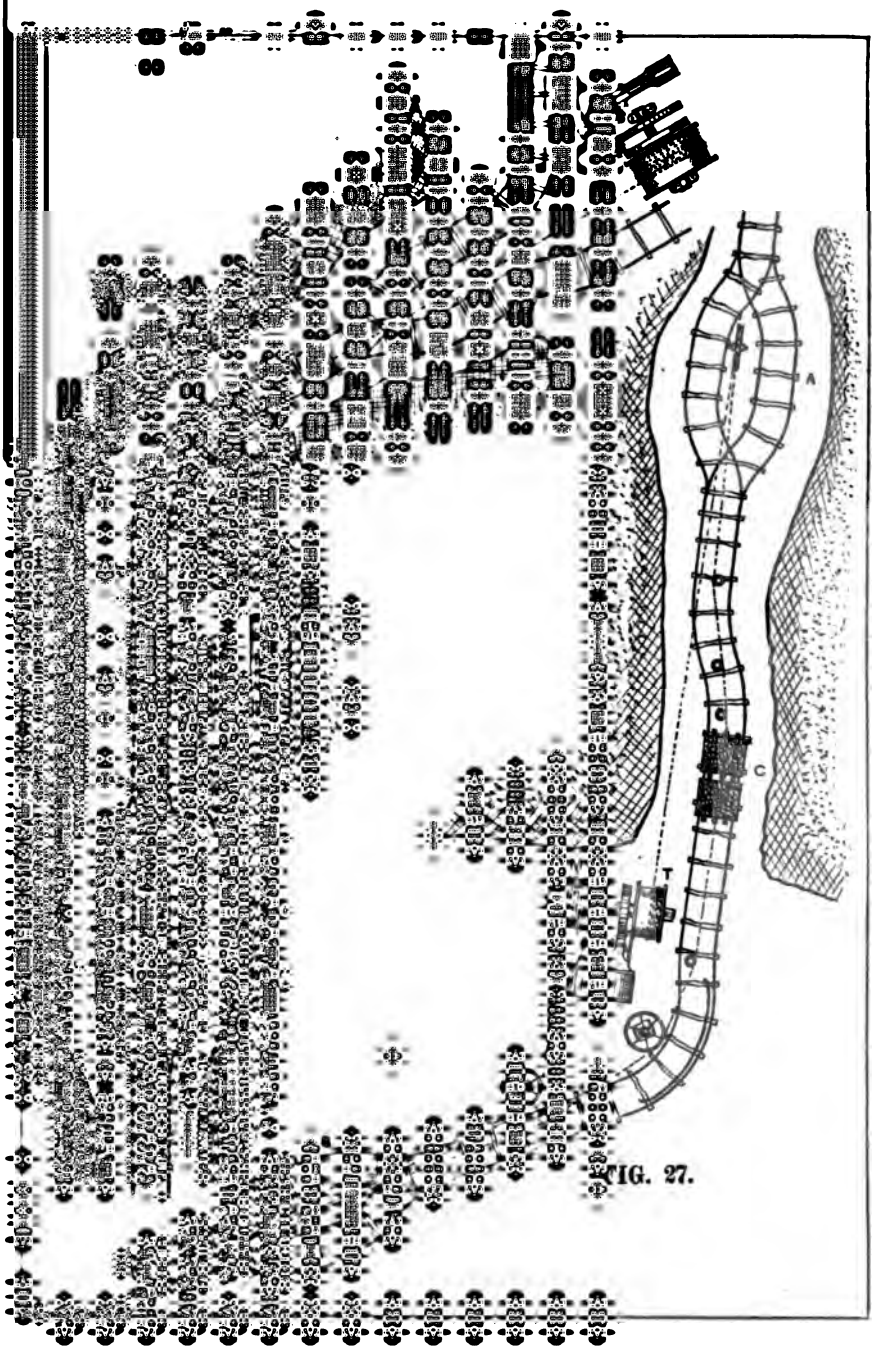


FIG. 27.

II. c. AERIAL PLANES.*—It frequently occurs that the character of the route renders the construction of surface planes a matter of great difficulty and expense. To such cases the application of aerial planes has peculiar advantages. These planes consist of wire ropes extending between the points of desired communication. Upon these cables the carrying vehicles or carriages travel, the ropes serving as tracks. This system, as in the case of surface inclines, admits of single or double planes, and may be operated by engine power or gravity, according to the conditions and requirements of the case.

Aerial inclines find frequent application in mountainous regions and in the hoisting of material from large open-cut mines or quarries. Fig. 24 shows a serviceable application of the single-rope system. This system is especially adapted to quarry work where the loading point is continually shifting. As shown, the incline consists of a single wire rope, one end of which is made fast at the bottom of the quarry, while the other end passes through the head of a derrick or mast to an anchorage on the bank beyond. The hauling rope, being fastened to one side of the carriage, passes downwards around the hoisting blocks and up again to the carriage, from whence it continues on to the hoisting engine. Attached to the main wire rope and immediately over the loading point in the quarry is located the stop block.† This block arrests the descent of the carriage beyond its position, but the hauling rope continuing to be paid out, the hoisting blocks are lowered into the quarry. Inclines of this character will readily hoist 150 to 200 tons per day.

Where the grades are excessively steep and large output is required, the double-rope system will be found very serviceable. The governing principles are similar to those of surface-inclines, the two ropes serving instead of the surface track. Figs. 25 and 26 show this system as applied to the diamond mines of South Africa.

III. WIRE-ROPE HAULAGE.

Experience has clearly demonstrated that, where the output of a mine is sufficient to warrant the use of mules in hauling the product

* I understand that Mr. E. Gybbon Spilsbury will present an exhaustive paper at the next meeting of the Institute upon the subject of wire-rope tramways. For this reason I will not particularize the details of this system of inclines, but simply mention some of the more important features. Mr. Spilsbury has also kindly examined the copy of this paper, and I am especially indebted to him for information in connection with aerial inclines.

† The loading-point may be changed by varying the position of the stop-block.

FIG. 31.

to the hoisting slope or shaft, the most economical method of underground transportation is obtained by the use of wire rope. So generally has this conclusion been accepted that, since its introduction a few years ago, the system has been extensively applied both at home and abroad. These remarks are especially true of the bituminous coal-regions of Pennsylvania, where there are but few mines of large output where one or another of the wire-rope systems is not in use.

Wire-rope haulage may be divided into three systems, viz.: the tail-rope, the counter-rope, and the endless-rope systems.

III. *a.* THE TAIL-ROPE SYSTEM.—In this system the hauling-engine is located at a convenient point, generally the adit, and the *main* and *tail* ropes are attached to two rope-drums, seated on the engine-shaft. The end of the main rope leading from the drum is attached to the forward end of a train of cars, while the tail-rope, running along the side of the entry to the end of the line of tracks, passes around a sheave or wheel, so located that the rope is brought midway between the rails. The rope continuing between the rails is fastened to the rear of the train. In connection with each drum there is a friction clutch, by means of which one drum may be driven by the engine, while the other runs freely on the shaft. Thus, when a number of cars are drawn to the adit the *main* rope is wound on its drum while the *tail* rope is paid out. To take the empties back into the mine, the ropes are attached to each end of the train as above, the engine is reversed, and the tail-rope, now acting as the hauling-rope, draws the cars back into the workings.

In application, this system naturally divides into two sub-systems: 1st, that in which the main entry only is operated by wire rope, the mining product being transported from the side-workings or entries to the haulage track by mules or gravity; and, 2d, that in which the side-entries also are operated by wire rope, the haulage-system in this case extending to all parts of the mine. One of the best systems of the former class is illustrated in Fig. 27, and is in use in one of the coal-mines on the Monongahela river. The circumstances of location are such that it is necessary to place the engine and drums at a point intermediate between the two terminals, and not, as is usual, at one or the other end. This makes it necessary to employ two tail-sheaves, one at the parting A, and another at the parting B. The main rope is fastened to the drum M, and passing off is led around the tail-sheave at B, and attached to the forward end of the train at C. The tail-rope, on the other hand, is fastened to the drum T, and is led thence around

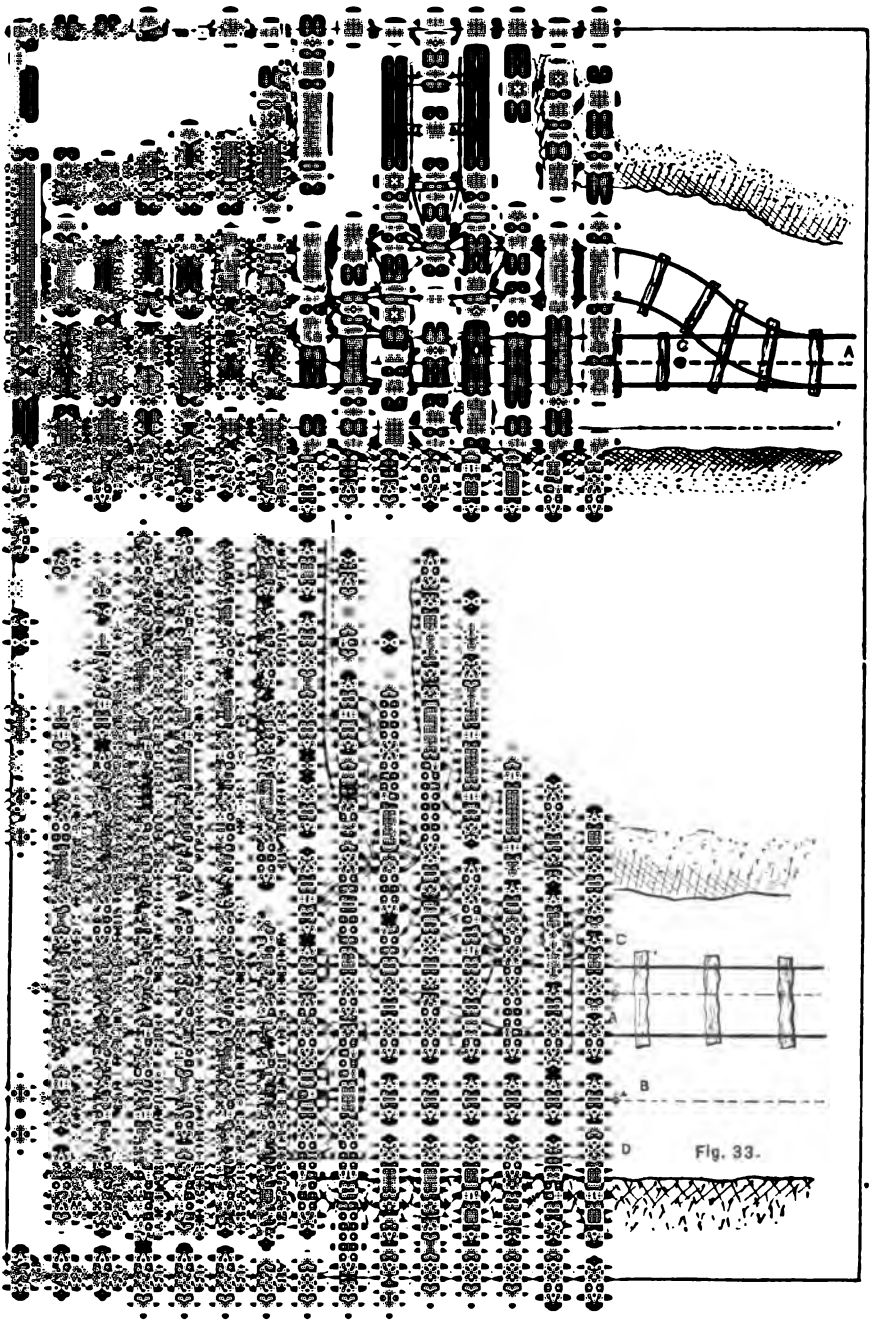


Fig. 33.

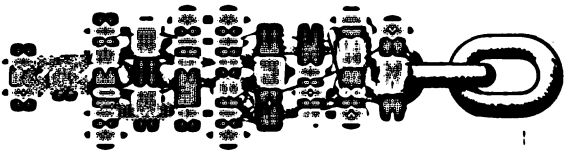


FIG. 34.

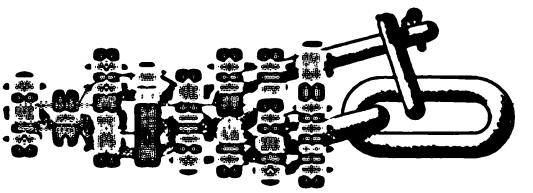


FIG. 35.

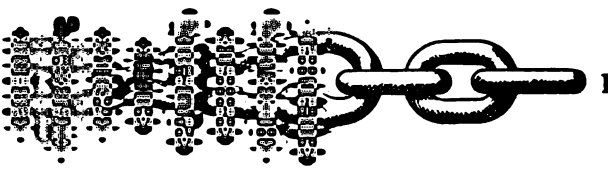


FIG. 36.

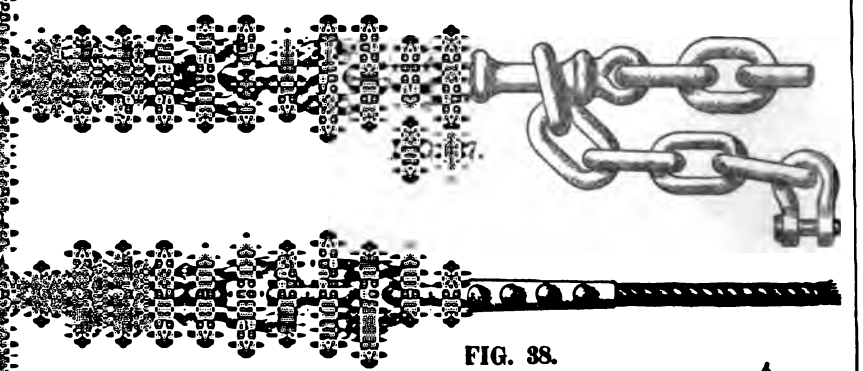
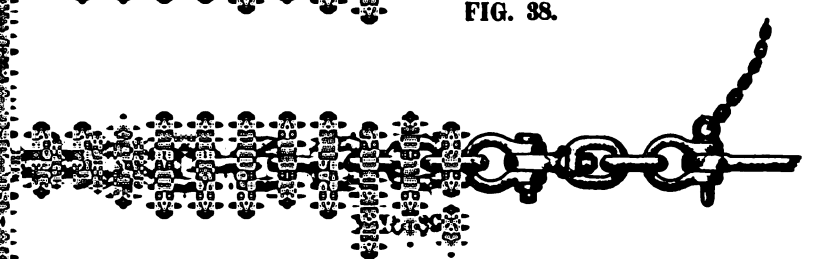


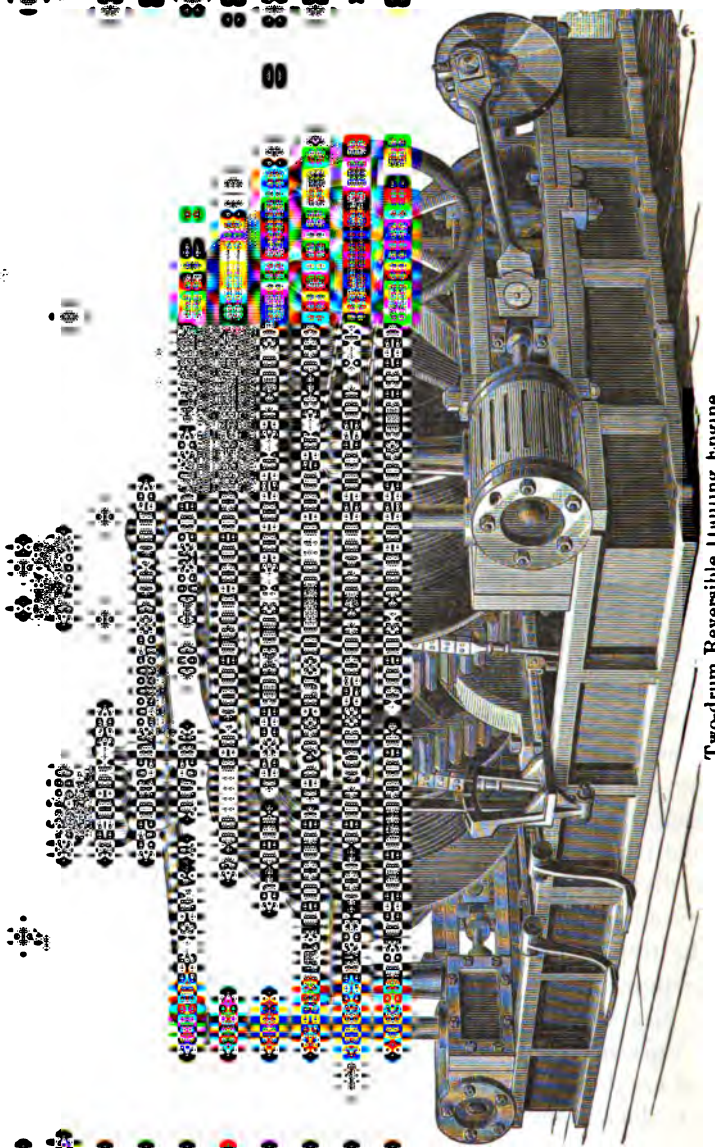
FIG. 38.



the tail-sheave A and attached to the rear of the train. The main rope is supported between the rails by 6-inch rollers, about eighteen feet apart; and the tail-rope is supported overhead along the side-walls by rollers supported from the hanging-wall or roof of the pit. The rope is guided around curves by 12-inch wooden sheaves, arranged as shown in Fig. 18. At the sharp bends, guide-sheaves 4 feet in diameter are placed, and guard-rails are introduced at the sides of the track to prevent the derailing of the cars due to the angular pull of the rope at this point. The engine is double-acting, with 10-inch by 12-inch cylinders, and 4-feet diameter drums, geared 1 to 5½. The road is approximately 3300 feet long, and its daily capacity is about 750 to 900 tons. The wire rope employed is steel, 7 wires to the strand; the main rope being ⅞ of an inch in diameter, and the tail-rope ⅝ of an inch. In this instance the coal is hauled by mules from the side-entries and branches to the general parting. When a sufficient number of cars have accumulated, the main rope is attached to the forward end of the train and the tail-rope to the rear end, and the train is hauled to the tipple. This is the simplest form of the tail-rope system, and were that system thus limited to the operation of one line, much less might be said in favor of it. To transport the mining product to the main entry with mules, and then transfer it to the rope-system, is less economical in every way than to extend the operation of the rope-haulage system to the various workings. The additional first cost of a complete system is very slight, but the advantages gained in economy and convenience are multiplied. Each entry is here provided with its own separate rope passing around a tail-sheave at the end of the working, and supported in the usual manner by rollers. The rope is of such length that the two ends reach to the main entry, and, when not in use, rest on the ground. Again, the rope in the main entry is divided into as many sections as there are entries or branches, the connections being arranged at such distances that when the train is at the tipple one set of these connections is opposite each side-entry or branch. In this manner, when it is necessary to haul a set of cars into any particular one of the side entries, the connections of the rope in the main entry opposite the side-workings to be operated are detached, and the engine-end of the main rope is attached to the extremities of the side-rope which, as before mentioned, extends into the side working. When the connections have thus been made, we have what corresponds to the original continuous

e for the rope is located in the

e usual methods of arranging
ws the arrangements permit-



Two-drum Reversible Hauling Engine.

the cars are at the adit. In
a set of cars into the side-entry
B, while the entry rope-ends C

a continuous rope
trate other arrange-
g the rope connec-

Double Four drum Hauling Engine.

and not, as in the
n Fig. 30 the guide-
permit the free pas-

sage of the rope, and the connections being severed, as before, one end of the entry-rope is attached to the train and the other to the main rope; and in this manner the side entry is operated as shown. The dotted lines at A show the position of the cars while the disconnection is made, B showing the train after passing into the entry. Fig. 29 shows an arrangement whereby but one disconnection is made in the main entry-rope, namely at A, the head of train. The end B of the entry-rope is here attached to the car, and the end A of the main rope, being free, is drawn a little farther by the engine and attached to the end C of the entry-rope. In both of these latter arrangements, where the road is not level, the disconnection is, of necessity, made while the rope is under a severe stress, thus rendering disconnection a difficult matter. This is not true of the method illustrated in Fig. 33, as the ropes are here disconnected while the cars are at the end of the system and the rope is consequently free from stress.

When operating a side-entry by either of the arrangements illustrated in Figs. 29 or 30 it is customary to facilitate disconnection by employing what are known as "knock-off" links. Figs. 34, 35 and 36 illustrate three devices that have given great satisfaction. Fig. 38 shows a shackle-joint, which is frequently used in making the connections between the sections of rope.

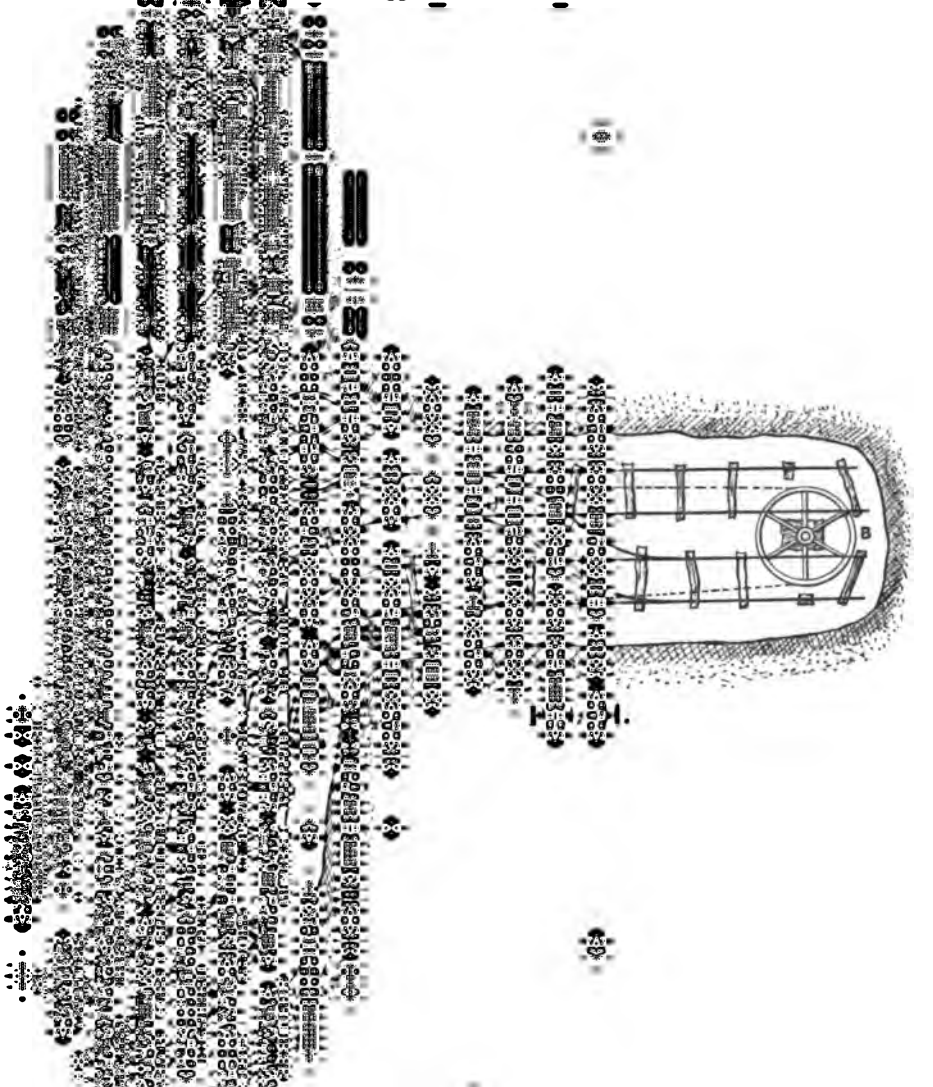
The usual method of locating the tail-sheave is illustrated in Fig. 31.

When it is found necessary to locate intermediate loading-points in the main or side-entry between two terminals, the arrangement illustrated in Fig. 32 will be found convenient. Suppose a branch to a side entry at X and it is found necessary to have a switch-station for loaded and empty cars at X. The empty cars coming from the direction A, the rope ends C and D are knocked off at these points, and the empties run on siding E. The rope-extremities are now about opposite the front and rear cars of the loaded train at F, and can be attached at the forward and rear ends and hauled to the adit. Other dispositions of tracks and switches are sometimes employed, but none that have proved themselves so efficient.

The rope is taken around curves by either of the two methods illustrated in Figs. 17 and 19. The main rope is supported as usual by rollers placed between the rails, and the tail-rope by rollers along the side of the entry, such as are shown in Figs. 15 and 16.

Figs. 40 and 40*a* show the improved types of tail-rope hauling-engines manufactured by the Ottumwa Iron Works, Ottumwa, Iowa.

le two-drum engine.
angles, coupled to a
turn with two spur-



o drums, one for the
of engine is adapted
relative position of

the workings, to place the hauling engines in the center of the mine and to operate the rope-system in opposite direction from that point. To such conditions the engine illustrated in Fig. 40 *a* is admirably adapted. Four rope-drums are provided, enabling the operation of two separate and distinct tail-rope systems.

The motion of the drum in both the foregoing engines is controlled by an improved cone friction-device, the operation of which is simple and effective. These engines are also provided with automatic indicators, the purpose of which is to enable the operator to determine the position of the train at any moment, and consequently to govern all stoppages, and provide for variations in grade. I append a table of the tractive power of these engines.

Tractive Power of Hauling Engines (Approximate).

Horse Power.	Cylinders.		Pipes.		Drum Specifications.					Load in tons of 2000 lbs., on a grade of		
	Diameter in inches.	Stroke in inches.	Diameter of steam. Inches.	Diameter of exhaust. Inches.	Diameter in inches.	Length in inches.	Capacity of each in feet of rope.	Diameter of main rope. Inches.	Diameter of tail rope.	10 per 100.	20 per 100.	30 per 100.
40	8½	12″	3	3½	36	36	2500	[illegible]	[illegible]	15	7½	3¾
50	10	12″	3	3½	42	36	3400	[illegible]	[illegible]	19	9½	4¾
60	10	16″	3½	4	44	36	4000	[illegible]	[illegible]	25	12½	6¼
80	12	16″	3½	4½	48	36	5000	[illegible]	[illegible]	30	15	7½

III. *b.* THE COUNTER-ROPE SYSTEM.—This is closely allied to the tail-rope system, the chief distinction being that in this case the tail-rope is replaced by a main rope operated by a separate engine. Briefly, the counter-rope system consists of two stationary engines, one in the mine and one outside, alternately pulling a train of cars in and out by means of two ropes, the one attached to the forward and the other to the rear end of the train. One rope is thus wound over the hauling-drum while the other runs freely off its drum, which, for the time being, is disconnected from the engine. The details of this system being almost identical with those of the tail-rope system, I content myself with describing in brief a counter-rope plant in actual practice.

Fig. 28 illustrates the general plan of a mine in the Monongahela region. The main hauling-drum and engine are located outside the mine near the adit, while the other engine and drum are located about 600 feet from the parting at the end of the entry. The rope

is guided around the curve by six iron sheaves 18 inches in diameter, placed outside the track (see Fig. 18) about 10 feet apart. Guard-rails are, as usual, placed at this point to prevent the cars from leaving the track. The ropes in this plant are connected end to end by means of a chain, as shown in Fig. 37. The number of these connections varies according to the length of the train. The maximum number of cars hauled by this plant is 40, representing about 80 tons; in this case four connections are made. The length of the line is about 6600 feet, and the rise about 1 in 50. A crucible-steel rope, $\frac{5}{8}$ of an inch in diameter, is employed, and lasts about one year. The rope is supported every 20 feet by 6-inch rollers. The tail-sheave is 48 inches in diameter. The main drum is 60 inches in diameter, and is operated by a 16-inch by 24-inch engine geared 1 to $2\frac{1}{2}$. The fly-wheel weighs 5 tons.

This system has been extensively applied in Europe, the conclusion of European engineers being that the counter-rope system forms the cheapest method of underground transportation where the output exceeds 800 tons per day. This is undoubtedly true where only the main entries are operated, but if we include the working of side-entries the above conclusion is probably at fault.

III. *c.* THE ENDLESS-ROPE SYSTEM.—Endless-rope haulage may be divided into two systems: 1. That in which the motion of the rope is continuous in one direction; and 2. That in which the rope is driven alternately in opposite directions.

The first system is that known as the "double track." In this case one track is used for ingoing and the other for outgoing trains. The rope is led from the engine-drums to the center of one track and continued to the terminal, where it passes around a tail-sheave placed at the end of the line. It is here turned into the other track, and so returns to the engine. In the latter, or "single-track" system, the hauling side of the rope passes from the drums midway between the two rails to the terminal, where it is led around a similar sheave, and continued back to the engine along the side of the entry. The latter is practically the tail-rope system, used in connection with an endless wire rope. The chief advantage of the endless system is that it permits stopping or starting the train at will without interrupting the motion of the engine, and in this manner does away with the system of signals necessary to a tail-rope plant. A familiar application of this system is that of cable-railways for passenger traffic, which in fact is a complete and extensive endless-rope haulage-system.

In this system, as in the tail-rope, curves are passed most advantageously by the use of small iron wheels 12 inches to 24 inches in diameter, placed as shown in Figs. 17 and 19, their distance apart being governed by the radius of the curve.

The operation of the side-entries and branches in the single-track system is similar to one of the tail-rope methods (see Fig. 29), except that in this case the rope is continuous. Fig. 41 represents the method of operating a side-entry in connection with the double-track system. Let us suppose a train of empties at X. Should it be desired to run them into the entry B the switches are set at the branch D, and the grip being attached to the moving rope,

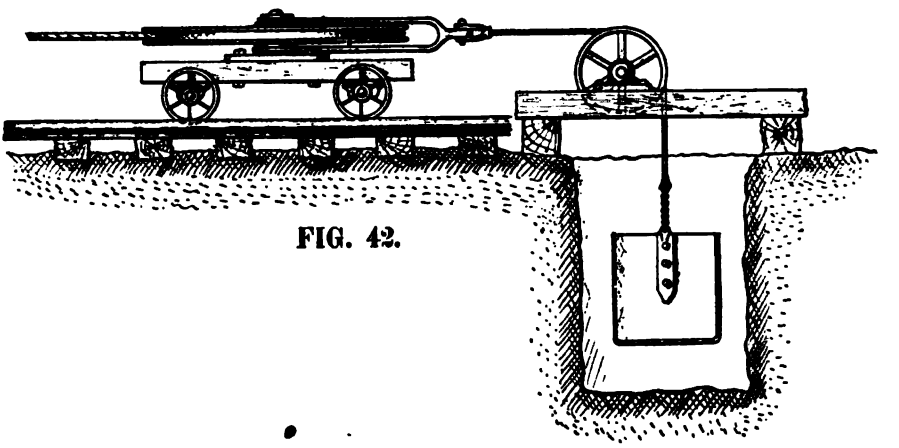


FIG. 42.

the cars are taken to F, where they are loaded and switched to the other track by F. Here the rope is gripped and the cars are taken to the junction with the main entry at K, where the rope is detached from the train, the latter being allowed to continue by its own momentum to the return track Y. The rope is again taken hold of, and the cars are carried to the adit. Should it be desired to run the cars direct to A the cars are detached from the rope just before reaching the entry B, and pass this point by their momentum; the rope is again gripped and the cars are drawn to switch E.

In this system it is of course necessary that the rope be kept taut, and for this purpose artificial tension is introduced. This is usually obtained by passing the rope round a wheel and supporting the latter upon a carriage to which is attached a weight, hanging over a pulley by a chain or rope. Fig. 42 illustrates this

arrangement. The rope varies somewhat in length on account of changes of temperature and elongation due to working under tension, and this device permits such variations without detriment to the tractive force or to the rope. An additional tension device is usually applied to the engine-drums, as will be shown hereafter.

Much time and ingenuity have been employed in devising convenient grips for attaching cars to the moving rope, but those which have been found the most serviceable in mining operations are of the simplest character and among the first employed. Fig. 43 illustrates a grip which may be employed when it is desirable to start and stop the cars without interrupting the motion of the rope. Two levers, B and C, acting on a pivot, K, with jaws, D, for gripping the rope, are actuated by the link A. In order that the grip may pass over the supporting rollers, it is necessary that the rope, when gripped, be elevated above the rollers. To this end, the U-strap, F G, is attached to the bumper of the car, and the grip-levers B and C, are provided with lugs H. A chain E forms the link between the grip and the car, by means of which motion is transmitted to the car. In the illustration I have supposed the grip in its lowest position, about to seize the rope. The operator, standing on the bumper at the front of the car, now presses the lever A and closes the jaws D. The rope, moving in the direction of the arrow, carries the grip with it until the chain E is taut. By this horizontal motion of the grip the lugs H, resting and moving on F, are raised to the level of G, and the grip and rope are likewise elevated. In this manner the supporting pulleys may be passed without difficulty.

Where starting is performed at stated points the following device may be employed. A grip similar to the foregoing is fixed by a bolt at K to the bumper of the car. The jaws are about $1\frac{1}{2}$ inches above the level of the rope when resting on the supporting rollers. In order to enable the grip to clasp the rope when starting a train, counterweighted supporting rollers are placed at convenient points. These pulleys raise the rope at the desired points to the level of the grip, and their vertical motion permits the passage of the grip without difficulty. Referring to Fig. 41 it will be seen necessary to introduce these pulleys at points H and Y. In many mines it has been found more convenient to adopt such arrangements that the rope is stopped for attaching and detaching. For this purpose I would recommend the device illustrated in Fig. 44.

When there are no side-entries to be operated it will be found

illy." The most approved
mperial mine in the Monon-
tical double-grooved drums,
ne-tracks. The hauling-rope
ver the forward drum and

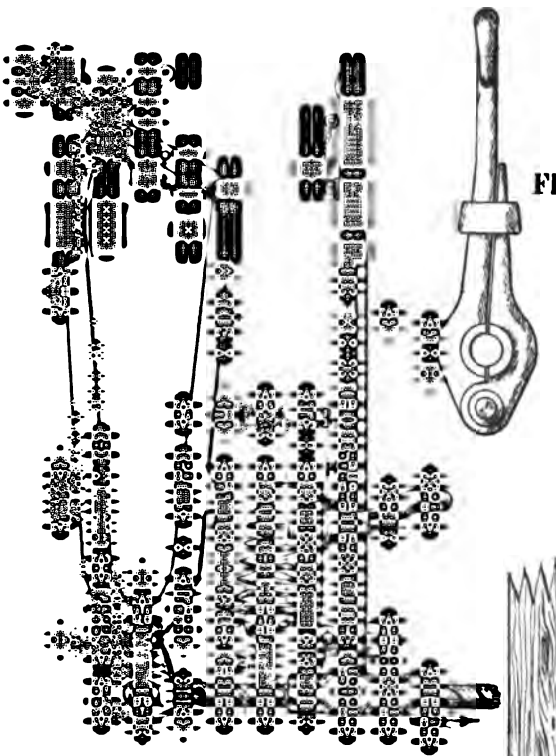


FIG. 44.

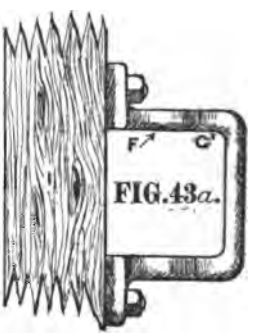


FIG. 43*a*.

repeated and the rope then
on of the drums is governed
stationary the drums revolve
to impart motion to the dilly
coupling being made to the
ing rollers are placed in this
ually 6 inches in diameter and
single-track system is em-

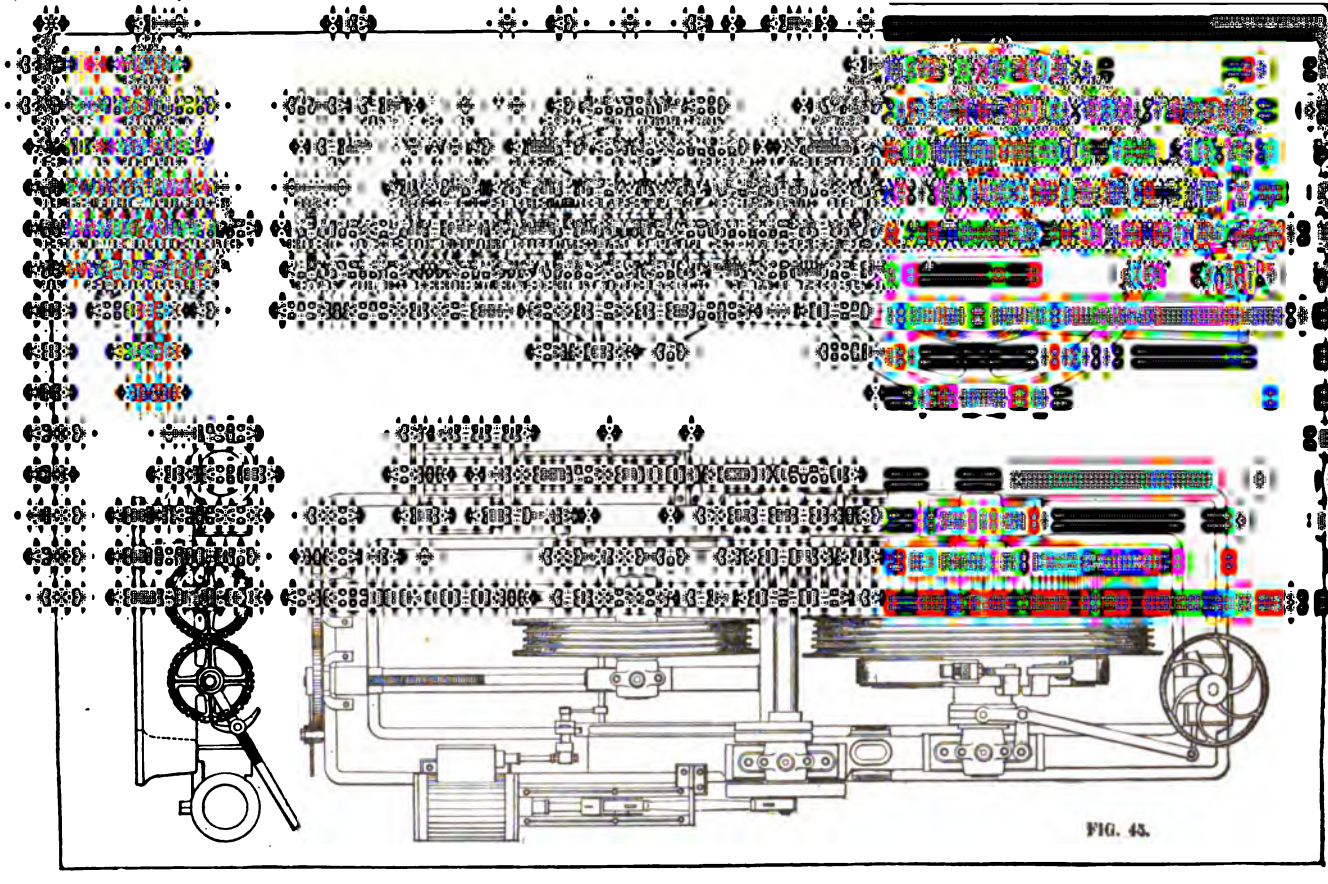


FIG. 43.

ployed, the rollers supporting the loose rope are of the same diameter, but from 6 to 12 inches in length.

Fig. 45 illustrates what I consider the most approved design for an endless-rope hauling-engine. It will be seen that there are here two drums placed tandem. The rope is first passed over the larger and then over the smaller drum, as many times as may be found requisite to secure the necessary tractive force. The motion of the larger or hauling drum is controlled by a friction-clutch. The motive power is furnished by two engines, actuating a pinion meshing with a spur-wheel keyed to the drum-shaft. The "idle" drum rests on sliding bearings, and slack rope may be taken up by draw-bolts as shown.

In proportioning the size of wire-rope requisite for a haulage-plant, the factors mentioned under the head of hoists and inclined planes must be taken into consideration in this connection also. In the tail- and counter-rope systems, the stress upon the rope is direct, and is due simply to weight and friction. In the endless-rope system, however, we have another factor to consider, namely, the amount of artificial tension introduced to prevent the slipping of the rope upon the drums. The tension depends, of course, upon the number of times the rope is passed over the hauling drum. In the case of the machine illustrated, where the rope has four half-wrappings around the drum, it is necessary to allow 20 per cent. more stress in the rope than in the case of the counter- or tail-rope system.

Much surface-wear is encountered in this application of wire-rope; and for this reason it is advisable in all cases to employ the best steel wire rope, composed of six strands of seven wires each laid about a hempen center. The table on page 238 gives the various data concerning this class of rope.

Conclusion.

In designing a wire-rope plant of any description, it is necessary, in order to obtain economical results, to proportion the details of the system in accordance with results derived from theoretical calculations. The engine-power, the sizes of ropes, drums, supporting rollers and curve-sheaves, as well as the position of the latter with reference to each other and the line of the road, should all be based upon direct calculation. The scope of the present paper does not

permit a discussion of the mathematical principles involved; but it may be declared that observation and experience have in every case supported these theoretical investigations.

Wire-rope has taken, by its own merits, a very important part in the development of the mining resources of the United States; and it is to be hoped that the few pages here submitted may contribute something towards a still further advance in the direction of economical mining operations.

MATTING DRY AURIFEROUS SILVER-ORES.

BY W. L. AUSTIN, TOSTON, MONTANA.

(Utah and Montana Meeting, July, 1887.)

THE only essential difference among the three methods of collecting the precious metals from their low-grade ores by fusion is comprised in the nature of the vehicle in which those metals are concentrated. If lead- or copper-ores are used, the processes are respectively lead- or copper-smelting; but when the sulphide of iron forms the matrix in which the gold and silver are collected, the process is known as pyritic smelting. The latter method presents rarely any advantages when lead- or copper-ores are available at rates which permit profitable treatment.

Pyritic silver-smelting, defined by Percy as "the smelting of silver-ores, which are either free from lead or do not contain it in sufficient quantity to collect the silver, in conjunction with pyrites, in order to produce a regulus in which the silver may be concentrated," is not, as the name would imply, confined to ores of this metal alone, but embraces auriferous silver-ores, or even gold-ores simple. This method of collecting the precious metals from low-grade dry ores, employed at Freiberg by Barthel Köhler in 1585, does not appear, up to the present, to have been introduced into this country on a working scale. And yet the possibility of getting rid in a single operation of all the earthy or siliceous gangue of a low-grade ore and concentrating the precious metals in a matte without loss of lead (which includes gold and silver) by slagging, or volatilization, at the same time producing a highly siliceous slag, very nearly if not quite as clean as is usual in lead-smelting, certainly commends itself. In some parts of the country, owing to the com-

petition met with from the large smelters, established at railway centers, lead-ores (and by lead-ores I mean such as in a smelting mixture furnish a sufficient percentage of that metal to permit economic smelting) command prices that render their treatment *per se* by local smelters an unremunerative operation; the result being that lead-ores are drawn away to other establishments, leaving behind a greater quantity of low-grade ores not suitable either for the milling process or for concentration, yet which might be advantageously collected in an iron matte.

The absence of this one from among our metallurgical processes is partially due to the perfection of the American silver-mill, which, in the majority of cases where the above specified ores are met with, is peculiarly adapted to Western conditions, namely, expensive fuel and labor. But the fact may be also accounted for by a disinclination to depart from the established lead-smelting process, even when the lead in the ore itself must be handled without profit, or, as is sometimes the case, at a loss.

During the summer of 1884, while engaged on professional work in Montana, my attention was attracted by the large quantity of dry silver-gold-ores awaiting reduction. The available lead-ore supply seemed utterly inadequate for economic silver-lead smelting; besides, competition from outside had brought these ores up to such a figure as to render their treatment unremunerative, nor were many of the ores at all suitable for wet concentration. But to offset these disadvantages, extensive bodies of iron pyrites, within easy access of the dry ore, suggested the practicability of pyritic smelting. A close examination of the field led to the conviction that such a process was the best means of beneficiating the majority of these ores, namely, those high in silica and low in silver and gold, and to the building of the Toston smelting works.

Toston is a small collection of buildings on the banks of the Missouri river, about 40 miles east of Helena, on the line of the Northern Pacific, a point about equally distant, measured by cost of transportation, from the dry ores and the pyrites mines, and at the same time in close proximity to lignite coal and limestone quarries. The Toston plant was intended solely for experimental purposes, to determine how far the process was adaptable to Western mining enterprises, namely, to quick returns on capital invested, when coupled with those necessary adjuncts, expensive fuel and high-priced labor. At the time of starting these works, no information of the process having been tried in this country, beyond vague

rumors of attempts made in Colorado, could be had; therefore there was no American precedent to serve as a guide. To copy the European practice with its large percentage of fluxing-material, small furnace-capacities, and high fuel-consumption, was out of the question. The practice at Kongsberg, Sala, Lend, Freiberg, Schemnitz, Lower Hungary, Transylvania, and the Altai mountains, was carefully studied as far as available literature would permit; and in the spring of 1885 the first furnace was built, a rude affair constructed wholly of sand-stone, with water-tuyeres, and the approximate dimensions of the ordinary Western lead-furnace. The feasibility of the scheme was so conclusively shown by the operations of this furnace, that foreign capital took hold of the enterprise, and the experiments were continued with better apparatus. Different styles and forms of furnace were tried, including the Herreshoff patent, and the works were being fitted up for handling ore on a commercial scale, when the recent passage of the Alien act by Congress caused a suspension of operations, and resulted in the temporary leasing of the property to a company handling the newly discovered lead-ore of the Cœur d'Alenes.

The three years' work at Toston demonstrated that, under conditions existing at that place, the best style of furnace for the matting process is what the Germans call a *Spurofen;* that is, a furnace without crucible, which is closed a short distance below the tuyeres, and from which the smelted products are permitted to flow continuously. The many difficulties which presented themselves as long as a crucible-furnace was employed, disappeared altogether when that furnace was converted into a *Spurofen*, and an outside receiver was attached. A duplicate of the Herreshoff furnace, which does such excellent work at Laurel Hill, New York, was tried and abandoned, on account of the passage communicating between the furnace and receiver becoming constantly choked up. This may have been due partially to the tendency of iron matte to chill suddenly (its fusing point lying above both that of copper and lead matte), and partially also to the use of highly siliceous slags, which do not admit of rapid smelting, and consequently did not furnish a sufficient flow of molten matter from the furnace to the receiver to keep the passage open. On a basic charge and copper matte, the furnace is said to work perfectly. Narrowing the smelting-zone and depressing the tuyeres, also assisted operations materially.

On the continent of Europe, blast-furnaces are used almost altogether for pyritic smelting (raw smelting, as it is termed). At Sala,

in Sweden, the furnace used is a *Spurofen*, with a tendency toward the sump type; but the latter is the usual style of furnace adopted. They are generally built very high, of small capacity, with one or two tuyeres in the back; whereas, at Toston, the best results were obtained with a very low furnace and large volume of air. Considering the perfection which furnace-building in this country has attained, it should be possible with a furnace of large capacity (about 150 tons in 24 hours) to make very satisfactory returns.

The method of feeding a matting-furnace is of the very greatest importance, for unless precautions are used, the easily fusible sulphide of iron agglutinates the charge above the smelting-zone, and chokes up the furnace as fast as it can be barred off.

The slags made at Toston range in silica from 30 to 48, in protoxide of iron from 27.24 to 49.73, and in lime from 4.7 to 26.98 per cent. Numerous experimental slags were tried, the object being to run the silica up as high, and the lime as low, as consistent with economic work. It was found much more difficult to keep the silver than the gold out of the slag. Slags showing merely a trace of gold (duplicate crucible-assays, one assay ton each) were made when the matte carried as high as $67 per ton in that metal, and 125.5 ounces of silver. Although the silver would run up more often above than below an ounce, it did not, with the proper combination of silica, iron and lime, average as high as 2 ounces. According to Percy, the slag made at Kongsberg in Norway usually carries less than 1 oz. per ton (2240 lbs.) in silver on a matte running about 80 oz. per ton. Lime certainly has a very beneficial effect on the slag. Our cleanest Toston slags went high in silica and lime, and low in iron. As the losses are usually supposed to be mechanical, a siliceous lime-slag may assist the separation of the metal by reducing the specific gravity of the slag; but Percy's experiments in fusing Ag_2S with different substances, among which were iron and lime, suggests a possible chemical origin for these losses. These experiments show that where a substance, such as KHO, $NaHCO_3$, or metallic iron (?), decomposes Ag_2S at a high temperature, the resulting fluid product carries considerable silver, while a great portion separates out in the metallic form; whereas, when lime is mixed with Ag_2S and subjected to a white heat, no chemical action takes place. Kerl also recommends a bisilicate carrying lime. However, I got very good results on a slag carrying the highest percentage of iron and the lowest of lime mentioned above. Local conditions make the economy of such a slag more than counterbalance the loss.

I am not prepared to say how far the concentration of the precious metals in an iron matte can be safely carried. The richest car-load of matte shipped from Toston ran 3.35 oz. gold and 125.5 oz. silver (approximately $200) per ton; but Kerl speaks of concentrated matte made at Kongsberg, carrying from 583.33 to 729.16 oz. per ton (2000 lbs.), and my experience leads me to believe that a more valuable matte could be produced than that above mentioned as shipped by us. The loss both of gold and silver is almost wholly in the slag, and is usually explained as due to the mechanical adhesion to the latter of small particles of matte. The higher the temperature at which the separation takes place, and the lower the specific gravity of the slag, the better the separation and consequently the cleaner the slags. In Europe it has been the practice at most places where the pyritic process is used, to roast, concentrate and desilverize the matte by a complicated series of operations, all having in view the production of bar-silver. In this country, concentration and refining can be omitted from the calculations of the producer, as a ready market exists for the product at almost any of our large copper- or lead-smelting establishments. One offer for our Toston matte f. o. b. at works was $20 per oz. for gold contents, and 95 per cent. of N. Y. quotations on silver, deducting a treatment-charge of $15 per ton, and $1.50 for freight. For lead-bullion of the same grade, the best we could do, at the time inquiries were made, was, for silver, 97 per cent. of N. Y. quotations, and for gold the same as in the case of matte; but the freight-charges to market amounted to some $23.40 per ton.

Comparison of Cost of Shipping and Refining Bullion and Matte.

Value of material $200 per ton, half gold, half silver. Gold estimated at $20 per oz.; silver at $1. No allowance made for loss in handling lead.

	Bullion.	Matte.
Gold,	$100	$100
Silver,	97	95
	$197.00	$195
Less freight and treatment,	23.40	16.50
Total proceeds of one ton,	$173.60	$178.50

Showing a margin of $4.90 in favor of the matte. Owing to the low grade of material which is aimed at by the matting process, it is hardly probable that a matte will be produced in one operation carrying over $200 per ton. However, if circumstances should demand a concentrated product, the first matte could be partially roasted and smelted again with siliceous ores, putting the slag resulting from the operation (which would probably be too high to throw away) back into the first smelting.

Since writing the above, I have seen Mr. E. Gybbon Spilsbury's paper on "Experiments in Matting Iron Sulphides." Mr. Spilsbury has been misinformed in regard to my own work, to which he refers. We never purchased any copper-ores at Toston, nor have we purchased lead-ores during the past two years. Our best results were obtained when running on pyrites alone, with just enough siliceous material mixed with them to insure a proper slag. Our pyritic ores, although not requiring concentration, occupy a fissure in solid diorite (?), averaging about 30 inches in width, and are so broken up in mining that 50 per cent. has to be sent to the calciner and roasted, the material being afterwards mixed with lime and "bricked" preparatory to being put through the blast-furnace. The failure of the gold to concentrate in the matte in the Boston experiments referred to by Mr. Spilsbury, might have been due to the absence of any metal with which the gold could associate itself. In the Toston experiments, besides a little lead which usually separated out in the metallic state, the matte contained silver in varying amounts from 20 oz. per ton upwards. That the gold could not be kept out of the slags at Boston, might also be accounted for by the low heat to which the charge was subjected. The difference between the results obtained from a blast-furnace and those from laboratory experiments can be in a measure anticipated. I do not recall any slag made at Toston which carried as high as 0.2 oz. gold per ton, and the average was below 0.05 oz. We estimate that at Toston we can handle $13 gold-ores by themselves without loss, but when mixed with dry ores at ruling prices we can treat such gold-ores at a profit. This includes mining and hauling 12 miles, the use of coke at $19 per ton, and the shipment of the matte half across the continent for further treatment. Of course, with facilities for wet concentration a much lower grade of material can be handled.

Table showing the Average Value of Ores Treated by Pyritic Smelting in Other Countries.

At Lend, in Austria, in 1866:

Gold,	0.117 ounces per ton.
Silver,	9.915 ounces per ton.
Copper,	2 per cent.
Lead,	1 per cent.

At *Kongsberg*, Norway: silver, 6.71–35 ounces per ton.
At *Zalathna*, Transylvania: 9.65 ounces auriferous silver per ton.
In the *Altai Mountains*, Russia: 14.58–17.5 ounces silver per ton.
In *Lower Hungary*: 17.50 ounces silver per ton.
At *Sala*, in Sweden: 8.46 ounces silver per ton.

Discussion.

Dr. Thomas Egleston, New York City: I have listened to Mr. Austin's paper with the greatest interest. The problem of concentrating the precious metals of poor ores which contain neither copper nor lead is a new one in this country, and has not, to my knowledge, been attempted before on any very large scale, either because of the price of fuel, or because the distinctly American methods of treating ores offered, for the time being, an easier solution. But when there is neither copper nor lead to be used as a conveyor of the gold and silver, the question is a pressing one, and Mr. Austin has suggested an answer which, from the careful method of investigation which he has presented to us, I have little doubt will be satisfactory. The problem how to concentrate silver and gold in an iron matte, seems at first sight a very simple one. Assuming that there is no other metal to carry off the gold and silver from the iron matte into a metallic regulus below, the problem seems to be to produce an iron sulphide which will contain all the gold and silver, and a siliceous slag so fluid that all the grains of matte will settle out of it, and thus to get rid of all the gangue in a single operation and without producing any chemical change, and thus to concentrate the precious metals. To do this, however, is not so simple. In copper and lead there are either the affinities of the metal or chemical reactions which can be depended upon to produce the concentration; but, although the largest part of the silver and gold found in the world is in some way either combined or associated with sulphur, we do not yet know of any reactions tending to concentration in an iron matte, other than the action of gravity. If it can be ascertained

that a compound can be made of gold, silver and iron with some substance, which will be heavier than a compound of sulphur and iron, so that liquation can be effected in the iron matte, producing an alloy which will be so much more dense than the matte that the gold and silver will separate, forming what the English, using the reverberatory furnace, call "bottoms," the problem would be solved; but we do not know as yet of any such alloy. Mr. Austin seems to have solved the problem of simple smelting, and, if there is any such thing as the production of iron bottoms in a shaft-furnace, will probably find the method of producing them. Concentration by fusion, which he suggests, is also possible, but it will probably be cheaper to send the first matte to market than to concentrate it by fusion. The use of the *Spurofen*, while it seems to have the advantage of avoiding engorgements in the furnace, is really only the production of a matte by liquation. It would have been very interesting if Mr. Austin could have given the details of the experiments which led him to the adoption of this furnace, and the abandonment of the *Sumpfofen*, which seems the natural furnace for this process, and what attempts were made to overcome the difficulty of too high a heat for the iron sulphide and too little for the slag. In this case there was no necessity of concentration, but, if it could have been done, the richer product would have been more marketable, while the poorer products could have easily been retreated.

Mr. Austin has not touched another subject which would have been of the highest interest to us all, and that is, what the relation of loss by volatilization is in this process to that of the other processes. I have had occasion to point out* that, in the ordinary lead-smelting, silver might be carried more than a mile without condensation when melted in the presence of considerable quantities of lead containing both arsenic and antimony. The complete absence of these last two metals, and the small amount of lead, will probably make these losses much less, and it would have been very interesting if we could have known what they were. Losses, due to improper chemical constitution or pasty condition of slag, have been long studied, but there is too much taken for granted. Slag-assays rarely show the total loss, and I have very little doubt that the search for the difference would throw much light on the proper method of conducting the process. It is to be hoped that these experiments will be continued, and that others will try the same method with the same energy and ability which Mr. Austin has shown.

* *Transactions*, vol. xi., p. 379.

E. G. SPILSBURY, New York City (communication to the Secretary): Mr. Austin's paper is certainly one of intense interest to all of us who are, in any way, wrestling with the difficult problem of beneficiating low-grade auriferous pyritic ores. Too little attention has, I think, been heretofore paid to the possibilities of economical treatment of such ores, by processes such as that described and practiced by Mr. Austin.

We must not lose sight of the fact, however, that the introduction of this system of matting, or what may be termed "igneous concentration," so far as it is at present developed, would be only economically possible in certain districts and on certain ores. Its success depends fully as much if not more than that of water- or air-concentration on the difference of the specific gravities of the materials treated. We all know that, generally, the gold in sulphides is in an infinitesimally fine condition; and, therefore, in order to permit its particles to sink in a molten mass by their superior gravity, it will be not only necessary to have an excessively fluid slag, but some agent must be present which will amalgamate with and so agglomerate the gold particles as to permit them to sink by virtue of their superior gravity through the molten slag. In other words, no separation of the metallic particles from the slag is possible, unless these particles are of such a size, that the difference of their specific gravity from that of the surrounding fluid is enough to overcome the friction of the fluid on the surface of the particles, which tends to hold them in suspension. Iron alone does not appear to amalgamate readily with gold. It, therefore, seems to me that the process necessarily requires the presence of either silver, copper or lead to make it thoroughly successful. Mr. Austin, as I understand it, himself admits this, and thus corroborates the results obtained by myself in experimenting on plain auriferous pyrites.

To be able to run successfully in a blast-furnace, it would also be necessary that at least a large proportion of the ore should be in lumps—a condition which would probably preclude the economical application of Mr. Austin's process to the treatment of fine auriferous iron sulphides, resulting exclusively from stamp-mills or concentration works.

Still, notwithstanding the possibly only local application of the system to certain districts, the fact remains that Mr. Austin is entitled to the credit of having initiated and perfected what is really in this country a new metallurgical process, that of matting auriferous silver-bearing iron sulphides, and so has made possible the

treatment by fire-concentration of certain deposits in Montana of low-grade gold- and silver-ores which could not heretofore be utilized with any profit.

C. KIRCHHOFF, JR., New York City (communication to the Secretary): Mr. Austin's practice at Toston is valuable, as showing that under special circumstances matting pyritic silver- and gold-ores may prove the most economical. In the absence of a comparative statement of the costs from ore to marketable product with the method adopted and with lead-smelting, purchasing ores, it is difficult to draw any inferences how widely, territorially, matte-smelting is applicable. One point indicates that circumstances were locally more in its favor than is likely to be the case in any but isolated instances. It will be noted that the freight-charge is only $1.50. On the other hand, Mr. Austin, in the case of base bullion, throws the entire treatment and freight-charges on the precious metals, when naturally the lead bears a part of the burden. This method of computation is fair only when it can be proved, as, indeed, happens in some cases, that the price paid for the lead in the ore is equal to or even greater than can be realized for it as bullion.

MR. AUSTIN (communication to the Secretary): The inference drawn by Mr. Spilsbury, that the success of this process depends wholly upon the difference in specific weights of the various bodies involved, is, in the main, correct; yet only so far as the same words might apply to any other smelting process.

That the gold contained in iron pyrites is in the metallic state, divided up in infinitesimal particles, and is collected out of a viscous, highly siliceous slag by alloying itself with the one or two per cent. of silver, copper or lead contained in the matte, is improbable. The minute particles of gold could not separate in the metallic state out of a thick oily slag in the short period allowed them for such a purpose; and as the other metals mentioned above go into the matte in combination with sulphur, in the form of sulpho-salts, there remains no metallic body with which the gold could alloy itself. Hence, it seems possible that it also enters the matte chemically combined.

Gold builds sulphur salts with the alkalies in the same way as does silver. Silver goes a step farther, and combining with sulphur and iron, gives us the mineral Sternbergite ($Ag_2S + 2Fe_2S_3$). May there not also be a combination of sulphide of gold and sulphide of iron, or some other metallic sulphide? Unless some such explanation is admissible, it is difficult to understand why the collection of the

gold in the matte should be comparatively more perfect than that of the silver.

Bloxam (*Chemistry*, edition 1867, p. 403) writes:

"In smelting the ores of gold in Hungary, the metal is concentrated by means of sulphide of iron. The ore consists of quartz and iron pyrites (bisulphide of iron) containing a little gold. On fusing the crushed ore with lime, to flux the quartz, the pyrites loses half its sulphur, and becomes sulphide of iron (FeS), which fuses and sinks below the slag, carrying with it the whole of the gold."

This idea of fusing the raw sulphurets with lime has, in two instances at least, been tried on this continent, with results highly encouraging to further experiment.

In this connection one is reminded of the fact that the condition of the gold in iron pyrites has never been satisfactorily demonstrated.

As regards the smelting of fine pyrites, concentrates for instance, in a blast-furnace, this can be done by running the charge very low, as is common practice with copper-smelters in the West, when a strong reducing action is not necessary. The blast in such a case passes up through the charge freely, keeping the whole mass in a condition—for want of a better simile, I will say—like that of boiling water in a pot. Of course, dust-chambers are a necessity when this is done.

Or, the reverberatory furnace might be used for treating such fine material. At Freiberg, when matting was extensively carried on, the reverberatory was used in preference to the blast-furnace.

The adaptability of this process, assisted by modern improvements in furnaces and associated appliances, is capable of extension; and the available market for iron matte is of itself an inducement to investigate the matter further. Recently a treatment-charge was made us of $2 per ton on iron matte, with the possibility of even that small tax being stricken out. This amounts to practically paying for the iron-contents!

Investigations on several details which readily suggest themselves, connected with the subject under discussion, are being followed up; but on account of their incompleteness have not been touched upon in this paper. The results of this work, if successful (and there is much encouragement), I hope at some future date to lay before the members of the Institute.

In reply to Mr. Kirchhoff's comments I would say that the costs of producing matte or bullion are about equal; furnaces of the same capacity are used, requiring approximately the same amounts

of fuel and labor. The freight charges of $1.50 per ton referred to must be taken in connection with a treatment-charge of $15, making a total of $16.50. Another bid for this product was, freight $11.95 per ton and treatment $2; total, $13.95—which goes to prove that the matter of freight does not seriously influence the results given. As previously stated, lead-ores are at a premium in Montana.

F. W. CLARK, Boston, Mass. (communication to the Secretary): I think Mr. Austin has opened up a very promising field, and has practically demonstrated that under certain conditions gold and silver can be economically concentrated in an iron matte, making it possible to treat a certain class of ores not desired by the millman or lead-smelter. Our experiments on the Haile mine pyrite, sent to us by E. G. Spilsbury, were not at all successful, as will be seen by referring to his paper. They confirmed smaller tests previously made by him. The material operated upon was raw and roasted auriferous pyrite, containing 10 to 35 per cent. of silica, smelted without flux in a small reverberatory furnace (hearth 3½ feet by 3½ feet) and tapped into moulds. All our slags carried gold, from 0.5 oz. to 1.64 oz. per ton, and the matte contained from 1.2 to 8.5 oz. per ton. The slag carried no visible matte, except where it lay on the matte-cake. The slag probably contained matte in a dissolved state, like sugar in water; and the gold in the slag was, I believe, in a combined state, since repeated assays of clean slag would yield the same result. Mr. Austin's results confirm the opinion I formed at that time, that, for successful working, some other metallic sulphide than iron must be present in the matte, and that the slag must contain some other base than iron. I suggested at the conclusion of our experiments that we brick the ore with lime and smelt in the blast-furnace, making a slag containing 12 to 15 per cent. CaO, and expected much better results by this treatment; but owing to the total absence of limestone in the vicinity of the Haile mine, which would render the shaft-furnace and lime slag inapplicable in practice, this experiment was not tried. As Mr. Austin remarks, laboratory experiments are not conclusive; and I have no doubt that far better results would be obtained on a large scale with properly constructed furnaces, longer time, addition of a little lime, thorough rabbling, etc. But that slags sufficiently low in gold to be thrown away can be obtained, I doubt. Mr. Austin's matte undoubtedly contained lead, if metallic lead was reduced.

The fact that gold-bearing sulphurets are usually in a fine state

does not appear to be a serious objection, as a partial roasting is in any case necessary to obtain the iron oxide to flux the siliceous ore. After roasting, a certain amount of sulphate of iron is present, and if a little lime (5 per cent. or less) is added, and the mixture air-dried, it will be sufficiently coherent to stand a moderate amount of handling, and will work well in a low blast-furnace without giving an excessive amount of flue-dust.

If both siliceous ore and pyrite be fine, why not mix them, partially roast and slag, and then smelt in the blast-furnace? If the *Spurofen* type of furnace is used, and the slags are kept below 40 to 42 per cent. SiO_2, the smelting should not be difficult.

It has been suggested, since the experiments were made on the Haile concentrates, that, if after the charge in the reverberatory was thoroughly melted, a little raw iron or copper sulphide had been sprinkled on the charge, and the heat had been raised, it might have carried with it the gold in the slag.

TWO CONDITIONS OF PHOSPHORUS IN IRON.—SECOND PAPER.

BY BYRON W. CHEEVER, UNIVERSITY OF MICHIGAN, ANN ARBOR, MICH.

(Duluth Meeting, July, 1887.)

AT the meeting of the Institute held in St. Louis, in October, 1886 (*Transactions*, xv., 448), I presented a paper upon the above subject. From the well-known action of certain solvents on phosphide and phosphate of iron, the conclusion was then reached, founded on analyses, that phosphide and phosphate of iron must be present in all iron.

But this conclusion was based upon the reactions obtained in the analyses of iron phosphide and phosphate, in which the phosphorus existed in conditions different from those found in commercial iron. In the former, the phosphorus is concentrated; in the latter, diffused. This fact left a reasonable ground for doubting the existence of phosphoric acid in commercial irons.

Some direct method was therefore sought which would prove its presence. After investigating the subject for some time, I came to

the conclusion that this object could be attained by volatilizing the iron as chloride with pure chlorine, and analyzing the residue for phosphorus, which, if present, must be in the form of phosphoric acid; for the iron phosphide would be volatilized. Mr. H. M. Howe, and other members of the Institute whom I consulted, coincided with me in this opinion. Samples of iron and steel were therefore treated in a stream of chlorine, in the manner described by Mr. Watts (*Chemical News*, xlv., 279), and the residues were analyzed for phosphorus.

The residues from the same samples did not always present the same appearance or have the same composition: some were apparently free from oxide of iron, while others had a small percentage of it. I am of the opinion that, in those residues which apparently contained no oxide of iron, the phosphoric acid was in combination with the manganese. Some residues contained a large percentage of oxide of iron, without a perceptible increase in the percentage of phosphoric acids. These results were, however, rejected, except in the case of bar-iron, from which it was impossible to obtain any residue with a small percentage of oxide of iron. By examining column A in the annexed table, it will be seen that the pig-iron contained very little phosphoric acid; that the bar-irons contained from 90 to 100 per cent. of their total phosphorus as acid; that the decarburized metals varied much; and that the steels contained from 5 to 20 per cent. of their total phosphorus as acid.

I am unable to point out any cause for the variation in the steels; the difference in the percentages of phosphoric acid not being great enough to be attributed to the difference in composition.

For the purpose of ascertaining if there is any relation between the phosphorus in the residue after treating with chlorine, and that in the residue after dissolving in other reagents, I dissolved samples of iron and steel in copper sulphate, ferric-chloride, bromine, hydrochloric acid, and ammonio-cupric chloride, and obtained the results shown in columns B, C, D, E and F of the table. In the pig- and bar-iron thus treated, the amount of phosphorus in the residues is reversed; of the phosphorus in the pig-iron, little, if any, was found in the residue from treating with chlorine; whereas the residues from dissolving in weak acid and copper sulphate, contain about 70 per cent. The residues from dissolving in ferric chloride and ammonio-cupric chloride contain much less, which is due without doubt to the oxidizing action of these reagents, converting some of the phosphide to phosphate.

The residues from bar-iron similarly treated, contained little phosphorus. The residues from treating the steels in the same way contain about the same percentage of phosphorus as the residues from ignition in a stream of chlorine; therefore this phosphorus must be present in the form of phosphoric acid.

In dissolving the steels in dilute acids, I noticed that the phosphorus in the residues seemed to increase as the percentage of graphite increased, which led me at one time to think that what I had considered phosphoric acid might be a combination of phosphorus and carbon; but the results obtained from the gray pig-iron and the bar-iron disprove that theory.

In samples treated with a solution of ammonio-cupric chloride, until all of the copper was dissolved, and then filtered (without adding hydrochloric acid), all of the phosphorus was found in the residue, which contained more or less oxide of iron. The oxide apparently increased as the carbon diminished; the carbon, however, seemed to have no other effect than to retard the oxidation of iron.

That the phosphide of iron is oxidized to phosphate when the iron is dissolved in ammonio-cupric chloride, may readily be shown by washing out with water the flask in which the iron was dissolved, and then dissolving the oxide of iron which adheres to the flask with nitric acid and precipitating with molybdate solution.

The results obtained by the different modes of analyses employed, seem to confirm the theory of the existence of two conditions of phosphorus in iron.

The phosphoric acid is, without doubt, combined with a base or bases, in different proportions in different irons. In the steels, it is so combined as not to be acted upon by dilute acids, while in bar-irons the greater portion is dissolved. This may be due to the conditions of formation as well as to form of combination.

The phosphide of iron also appears to exist in two forms, the one being a definite form or combination, such as Fe_3P, which form has been shown by F. Schneider to exist in pig-iron, the other being in a state of diffusion. The former is not, and the latter is, dissolved by dilute acids. Pig-iron contains a larger percentage of the former than of the latter, while the steels analyzed contain diffused phosphide only.

Table of Analyses.

Variety of Iron.	No. of Sample.	Per cent. of				Per cent. of Phosphorus, calculated on the original sample, in the residue after					
						A	B	C	D	E	F
		C	Mn	Si	P	Ignition in a stream of chlorine.	Dissolving in copper sulphate as for carbon.	Dissolving in ferric chloride as for slag.	Dissolving in cold dilute hydrochloric acid	Dissolving in hot dilute hydrochloric acid	Dissolving in am.-cupric chl. as for carb.-test
Pig-iron	1				22	.0	.155	.048	.156	.156	.036
"	1				.22	.006				.150	
Bar-iron	1				.160	.168					
"	1				.1	.152					
"	1				.160	.169					
"	2				.135	.123	.073	.046	.029	.015	.012
"	2				.135	.120					
Decarburized Metal.											
Clapp-Griffith		.051	.0	.009	.157	.005			.017	.014	
" "		.051	.0	.009	.157	.002					
Bessemer		...	...		.090	.025				.012	
"		...	...		.090	.022					
Siemens-Martin		...	...		.109	.019				.017	.024
Steel.											
Clapp-Griffith	1	.13	.492		.42	.060	.048		.063	.040	.084
" "	1	.13	.492		.42	.055				.041	.090
" "	1	.13	.492		.42	.070					
" "	1	.13	.492		.42	.090					
" "	1	.13	.492		.42	.086					
" "	1	.13	.492		.42	.096					
" "	1	.13	.492		.42	.050					
" "	2	.16	.411		.175	.008	.027		.056	.031	
" "	2	.16	.411		.175	.010			.058		
" "	3	.10	.566	.008	.164	.014			.031	.016	
" "	3	.10	.566	.008	.164	.022				.014	
" "	3	.10	.566	.008	.164	.019					
" "	3	.10	.566	.008	.164	.010					
Bessemer	1	.35	1.10	.045	.108	.026	.016		.028	.013	.028
"	1	.35	1.10	.045	.108	.014			.028		
"	1	.35	1.10	.045	.108	.028			.036		
"	1	.35	1.10	.045	.108	.020					
"	2	.60	1.00		.110	.019			.029	.031	
"	2	.60	1.00		.110	.014					
"	2	.60	1.00		.110	.011					
"	3	.10	.65		.130	.027	.013		.047	.020	.034
"	3	.10	.65		.130	.018			.052		

H. M. HOWE, Boston, Mass. (communication to the Secretary): I have elsewhere pointed out that the existence of phosphoric acid

in ingot-metal, at least when this is made in siliceous vessels, is improbable on *a priori* grounds, for several reasons, and that the observed variations in the chemical behavior of phosphorus, and in its effects on the properties of this metal, are as fully and more simply explained by the existence of different phosphides of indefinite composition. Briefly recapitulated, these grounds are: 1. The slag which accompanies the production of the metal referred to is nearly, if not quite, free from phosphoric acid. 2. The ingot-metal is usually almost wholly free from this slag. 3. This slag is so acid that, even if phosphoric acid were intentionally introduced into it, it would be rapidly and almost completely reduced and absorbed as phosphide by the metallic iron.

Thus the conditions which permit the existence of phosphoric acid in weld-metal appear to be wholly lacking in ingot-metal, but besides this improbability, there is circumstantial evidence opposed to a belief in its existence. Phosphates contained in molten ingot-metal would reasonably be expected to gradually separate by gravity, whereas, in fact, phosphoric steel may lie for hours in tranquil fusion, without noticeable loss of phosphorus. This is, of course, not final; for it is quite conceivable that phosphoric acid or phosphates may be united to the metallic iron by some chemical tie which is strong enough to prevent separation by gravity.

These considerations, while certainly not conclusive, seem to me to argue so strongly against the existence of phosphates in ingot-metal, that unusually positive and unquestionable evidence is required to establish the fact of their presence.

In correspondence with Professor Cheever, I suggested the chlorine test. It seems ungracious to dispute results obtained by the method which I recommended; and I feel called upon to explain that, since recommending it, my attention has been called to a recently-published article, pointing out the absolute need of certain precautions, which have not in general been employed, and the great liability to error. This, coupled with certain internal evidence in Professor Cheever's results, compels me to reluctantly conclude that his praiseworthy labors, for which we must all thank him most heartily, and which I hope will be pushed till this interesting question is definitely settled, while they certainly lessen the improbability of the existence of phosphates in ingot-metal, do not suffice to change that improbability to a probability, much less to a certainty.

The internal evidence to which I refer is as follows:

1. Of the phosphorus present, the proportion which, according to Professor Cheever's table, appears as phosphate, is sometimes much smaller in unrecarburized blown metal than in that which has been recarburized. Now the addition of the recarburizer, according to all chemical analogies and the abundantly-published and carefully-studied phenomena of the basic process, should energetically deoxidize any phosphates present.

2. In one case 90 and in the other case 100 per cent. of the phosphorus of weld-metal appears as phosphate—a result so startling as in itself to raise grave suspicion.

3. The results obtained with mild solvents appear to conflict with those obtained with chlorine. To those who, agreeing with Professor Cheever's earlier paper, consider (as I cannot say that I do) that ready solubility is a strong indication of the existence of phosphates, this fact must further weaken the evidence of the chlorine test. The weak solvents indicate that most of the phosphorus of unrecarburized metal and of Bessemer steel exists as a phosphate; the chlorine method, that but little does. Again, the weak solvents indicate that much of the phosphorus of the cast-iron exists as phosphate; the chlorine test proves that none does. Here its evidence is well-nigh conclusive; for while, owing to experimental errors, the chlorine test may make phosphides masquerade as phosphates, the converse is improbable. This fact suffices, I think, to drive the weak solvents out of court as convicted perjurers. The weaker vessel deserves to be smashed.

To abandon the former ground, and hold that it is not difficult but easy solubility that indicates the presence of phosphates, helps little. Even in this view, the evidence of chlorine in the case of cast-iron completely breaks down the testimony of the weak solvents, for it here proves the absence of phosphate, while they still testify to its abundant presence.

If I may venture a suggestion, it is that Professor Cheever test his chlorine method by treating iron known to contain phosphorus wholly as phosphide and in other respects resembling commercial ingot steel as nearly as possible. To the analyst many methods of preparing such iron doubtless suggest themselves. For instance, precipitated iron oxide could be completely reduced in hydrogen and then mixed with a little iron phosphide and a minute quantity of iron silicate, the latter, because that present in commercial iron might be conceived to give rise to the formation of phosphoric acid during the chlorine treatment. Let this mixture, after protracted

sweeping with nitrogen, be fused in that gas or *in vacuo*, without removing from the apparatus, and then treated with chlorine. If no phosphate is indicated, and if, under identical conditions, phosphate is found in commercial iron, it would be hard to resist the evidence.

PROFESSOR CHEEVER: Mr. Howe, in closing his remarks, suggests the preparation of a small quantity of steel, which shall have a certain composition and be free from phosphates—in other words, contain phosphorus as pure phosphide of iron only. Such an undertaking would not, to my mind, relieve us of the difficulty, if such really exists, for a test would have to be made to prove the metal free from phosphate; besides, we have in cast-iron itself no phosphate, both theoretically and practically, as shown by the test with chlorine.

I do not deem it worth while to discuss any theories which may be advanced in regard to the behavior of iron and steel when subjected to the action of weak solvents. My views upon the subject are set forth in the paper under discussion.

Mr. Howe seems to think it a suspicious fact that I find 90 and 100 per cent. of phosphate in bar-iron. If there is an error in the work—and this is what he means to say—why do I fail to find phosphates in cast-iron?

But he is not quite consistent, for in the *Engineering and Mining Journal*, vol. xliv., p. 8, he says: "The presence of phosphates in weld iron is shown to be *exceedingly* probable by the fact that when it is volatilized with chlorine a *large* proportion of its phosphorus remains in the non-volatile residue." (The italics are mine.)

Again, the conclusions which he draws from the results obtained from the samples of decarburized metal are hardly warranted by the facts.

By examining the table it will be seen that the only sample of perfectly decarburized metal (namely, the Bessemer) has a much larger percentage of phosphate than any of the recarburized metals, namely, 27 per cent. The Clapp-Griffith and the Siemens-Martin samples were not perfectly decarburized, and there is no evidence here to show whether the recarburized metal of these samples contained more, or less, phosphate than the decarburized metal.

But why should decarburizing reduce phosphate to phosphide?

It has been stated that, by introducing lime with the blast, under certain conditions, phosphide of iron will be oxidized, forming phosphate of lime, the percentage of carbon remaining the same; therefore, if phosphate is formed during the blow, the introduction

of the recarburizer should have no effect, except, perhaps, to change the phosphate of iron to phosphate of manganese, for the phosphate of iron is not reduced as readily as phosphate of lime. It is also well known that if phosphates be brought into contact with molten iron in the presence of a highly acid slag, the iron takes up or absorbs the phosphoric acid. It is said that it reduces it to phosphide of iron. But what proof is there that the phosphate is wholly reduced to phosphide? Would it not be just as reasonable to say that part is reduced and part is absorbed as a highly basic phosphate of iron, which would have a specific gravity sufficiently high to prevent its ready separation from the iron? I do not claim that phosphorus is removed from the iron in the acid converter by being oxidized and entering the slag, but that a part is oxidized during the blow, and that it is not wholly reduced to phosphide by recarburizing.

While I differ with Mr. Howe as to the conclusions to be drawn from the results shown in the table, I feel bound to say, as he has said, that there are many difficulties attending the volatilization of the iron with chlorine. But I cannot say whether or not I employed the indispensable precautions to which he refers, since he has failed to state what they are.

I do not claim absolute accuracy for my results, but I do claim that the fact that I find almost no phosphate in cast-iron, and a large amount in bar-iron, which corresponds with the generally accepted theory, is good presumptive evidence that the results obtained from steel are near the truth, if not the absolute truth.

It may be that the phosphorus was oxidized to phosphoric acid in the chlorine process, but the fact that many of the residues contained no iron, and yet had the usual amount of phosphorus, is strong evidence against such a theory.

The only remaining point to which I wish to call attention is the intimation that phosphide may mask as phosphate. This, it seems to me, is completely negatived by the results obtained from pig-iron.

The apparatus used was so arranged that the chlorine passed successively through water, sulphuric acid, carbon heated to bright red, sulphuric acid and a condensing-bulb; and the air was expelled by the chlorine before the iron was introduced into the combustion-tube, so that the action of the chlorine upon the iron commenced immediately.

R. W. RAYMOND, New York City: In this connection, it may be interesting to consider a criticism of Professor Cheever's former

paper on this subject (*Transactions*, xv., 448), which has recently been published by Mr. Leopold Schneider, assistant in the Royal Imperial General Assay Office at Vienna. After giving Professor Cheever's results and conclusions, Mr. Schneider says (I translate from the *Oesterr. Zeitschr. für Berg- und Hüttenwesen*, July 30, 1887):

"As to the analytical method of determining the phosphorus, it is based by Professor Cheever upon the assumption that, when treated with cupric chloride, the small quantities of phosphide contained in the iron completely resist the action of cupric chloride, and, moreover, that all the phosphoric acid of the phosphate formed from the phosphide by the oxidation of glowing iron can be taken into solution by five minutes' treatment with a one per cent. hydrochloric acid solution, or even by the action of an ammoniacal solution of ammonium oxalate."

Both these assumptions Mr. Schneider disputes, citing experimental proof to the contrary, and concludes as follows:

"Cupric chloride, as the above experiments show, does at least have an observable reaction upon iron phosphide. The phosphoric acid thus produced, which goes into solution when the chloride is dissolved in alcohol, remains, in the case of an aqueous solution, with the iron, forming basic hydrated salts in the residue, from which it can easily be extracted with one per cent. hydrochloric acid. Since, on the other hand, the phosphoric acid of slag cannot be dissolved by that one per cent. acid, it is evident that the phosphoric acid found by Professor Cheever is mostly the product of the action of cupric chloride on iron phosphide. His results are, therefore, incorrect; and the explanation they offer of the different effects of given proportions of phosphorus in steel lacks trustworthy analytical proofs."

PROFESSOR CHEEVER: Mr. Schneider's remarks would be pertinent had they followed my first article, but, in this connection, are somewhat out of place, since what he says is, in the main, a repetition of the present article, and, for the most part, is not denied. He, however, makes some mistakes which I cannot pass unnoticed, as for instance, the statement that I dissolve with a one per cent. solution of hydrochloric acid or an ammoniacal solution of ammonium oxalate. I used a two per cent. solution of the former and a saturated —not ammoniacal solution—of the latter. Further comment is uncalled for.

H. M. HOWE, Boston, Mass. (communication to the Secretary): I must emphatically deny Prof. Cheever's charge of inconsistency.

Wrought-iron contains a small quantity of basic slag. Both this slag and the metal itself have a high affinity for phosphorus. Under these conditions, it is not surprising that the slag should retain a large proportion of the phosphorus as phosphate; but it would be, in my opinion, decidedly surprising if all the phosphorus should be thus held by the slag. Under such circumstances, phosphorus tends to distribute itself between metal and slag in proportions corresponding to equilibrium for the existing conditions. Only when the slag is extremely basic can it completely remove phosphorus from the metal. I have elaborated and supported this view on page 26 of the *Engineering and Mining Journal* for July 9th, 1887. The evidence to my mind is simply overwhelming, irresistible in kind and in amount. That Prof. Cheever's wrought-iron may have held slag so basic as to remove the whole of the phosphorus from the metal itself is of course quite possible, but sufficiently improbable to raise suspicion.

With regard to the decarburized and recarburized metals, I cannot agree with Prof. Cheever. His Clapp-Griffith decarburized metal, with 0.051 per cent. of carbon, I should consider practically thoroughly decarburized. Even prolonged overblowing sometimes will not reduce the carbon below 0.04 per cent. With 0.051 per cent., the metal would be almost certain to be oxygenated, and would certainly be much more likely to contain phosphate than recarburized metal; yet only about 2 per cent. of its phosphorus appears in Prof. Cheever's table as phosphate, while in his recarburized metals the proportion appears to be much higher: hastily running over the results, it appears to lie between 5.1 and 20.4 per cent.

That recarburizing should tend to reduce phosphate to phosphide appears to me too axiomatic to require demonstration. In the basic process recarburizing not only tends to, but often actually does, drag phosphorus from slag to metal, raising the proportion of phosphorus in the latter sometimes from .05 to .53 per cent.; and as the phosphorus thus acquired renders the metal as cold-short as any other phosphorus does, we reasonably infer that it is dragged back as such, and not as phosphate.

THE CONSTRUCTION OF MAPS IN RELIEF.

BY JOHN H. HARDEN, PHŒNIXVILLE, PA., AND EDWARD B. HARDEN, PHILADELPHIA, PA.

(Duluth Meeting, July, 1887.)

THE practical use to which topographical models or relief-maps have been put, has within the last few years taken a wider range. They are rapidly coming into favor for other than purely scientific purposes, and their value for illustration and instruction is being more fully appreciated.

Topographical maps, representing by contour-curve lines the elevations and depressions of the earth's surface, are now demanded in engineering work of nearly every description, and afford a ready means of bringing out the configuration of the country in a simple manner. A beautiful example of recent topographical mapping is seen in the seventeen sheets just issued by the Geological Survey of New Jersey.

By reason of the inability of unskilled persons to understand the nature of a topographical map, it is sometimes necessary to construct the same in relief, with the rivers, canals, railroads, etc., delineated upon it. With the aid of such a map, problems in engineering science which would otherwise require expert assistance to explain them, are reduced to a simple and understandable form.

Relief-maps are also of value to the geologist in working out the geological structure. To the instructor they present the most ready and efficient means of imparting clear and accurate knowledge of topography and kindred subjects. To the lawyer they are invaluable in enabling him to bring into the court-room a representation of a part of the country with all its features accurately shown, to explain to the court and jury the special points about which litigation is pending and which would otherwise require long and laborious description. The application of the topographical map to business purposes is shown in a relief constructed by the writers for the Cumberland Valley Railroad Company and used by that company in the various ways of calling attention to the desirability of its route. This model is based partly on actual survey and partly on sketch topography. The horizontal scale is $\frac{1}{80,000}$, the vertical $\frac{1}{19,200}$ of

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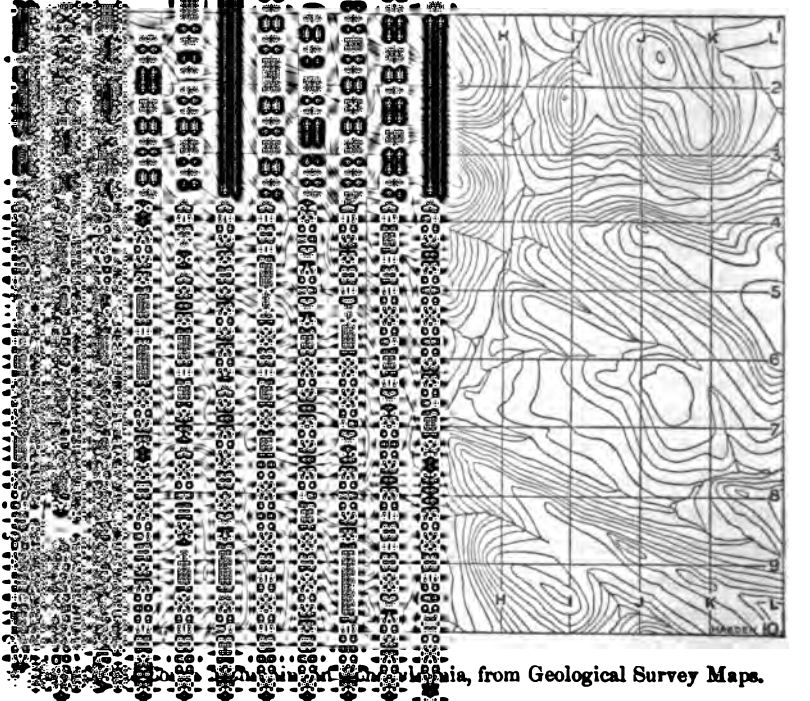

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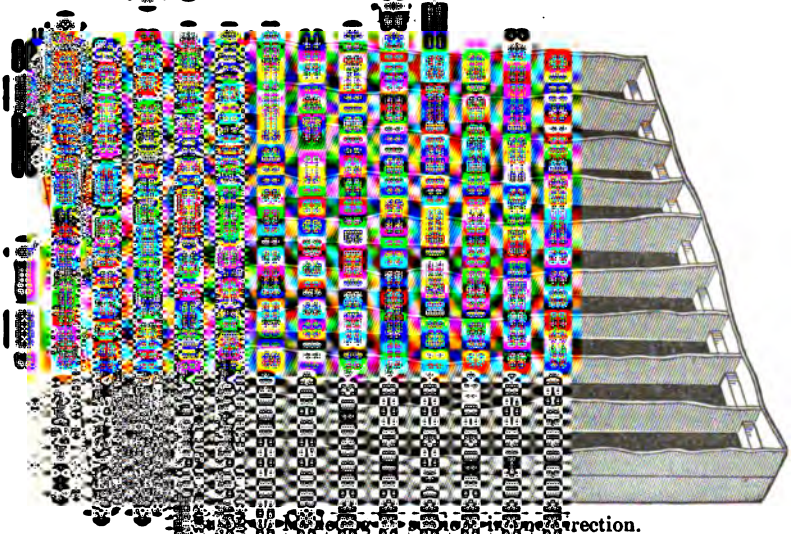

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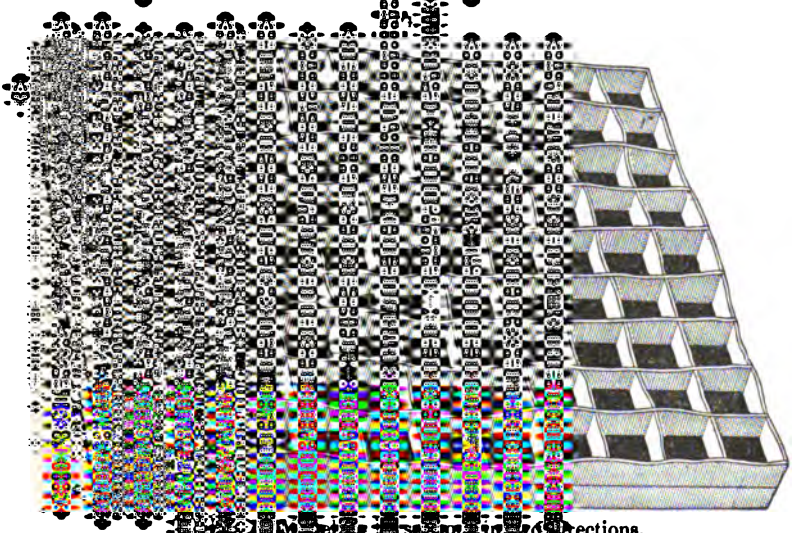

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point crossed by each contour, being measured from the base of the strips, and the profile thus obtained cut out. The position of these section-lines is transferred from the map to a baseboard, as a guide to adjust and fasten the profile-strips.

Fig. 2 shows the topography of the map (Fig. 1) drawn on profile-strips in one direction only. Fig. 3 represents the same area with profile strips also at right angles.

This method of making models was described in a work on Isometrical Drawing, by T. Sopwith (London, 1838), who says:

"Several sections being cut out in pasteboard, or, what would be better, in thin plates of copper so prepared with a covering of paint as to admit of being drawn upon with ink and colors, are to be joined crossways by what is termed half-lapping, that is, by cutting each section half way down where it crosses another section cut in like manner on the other edge, and the whole being properly painted with sections and joined together in this manner, will afford an interior view of the geological structure of the district. The model of squares thus formed is to be placed on a plane surface, and the several spaces may be filled with pieces of wood or Paris plaster carved or moulded on the upper surface so as to represent the surface of the earth. The several strips on which the sections are drawn may at any time be taken out for examination or for delineating any new discoveries upon them."

Mr. Sopwith published in 1841 an admirable series of small models to illustrate various geological phenomena connected with practical mining. They were constructed of various kinds of wood fitted together to scale from actual measurements of the strata in the North of England.

A method of construction for relief-maps, with or without the aid of contour-lines, is as follows. A baseboard is prepared, as in the method above described, and the contour-lines or points of elevation on the map are transferred to it. Nails are then driven into the baseboard at intervals and allowed to project above the board to the required height represented by the contours or points of elevation. The space between the nails is filled with wax or other suitable material, and tooled down to the heads of the nails, the finer detail being put in by reference to the map. This method may be used with advantage where the elevations are given at irregular intervals or where the topography is not shown by contour-lines.

The method adopted by Mr. E. E. Howell, of Rochester, N. Y., in the construction of a relief of the United States Geological Survey map in the vicinity of Leadville, Colorado, is thus described in Ward's *Natural Science Bulletin*, vol. iii., No. 2.

"The model is built up with cardboard representing twenty-five feet in thickness with the edges cut to the contour-lines. Theoretically this should work perfectly, provided the cardboard is of the proper thickness, but practically it will not. The sheets are sure to be closer in some places than in others, and just where they will do so is not always possible to tell, although as a general rule, small surfaces will lie more closely than larger ones. To meet this difficulty the main thickness of the model was built up of solid dry wood, which can be worked very exactly, and serves as a constant check in putting on the cardboard contours. After these are all on, the model will be finished with clay mixed with glycerine and then moulded."

A method used by Professor F. H. King, River Falls, Winconsin, is thus described in a letter to the writers.

"I have come finally to use a block of plaster of Paris, cast in a suitable frame of the desired size and thickness, as the best material to work upon. There are several methods of realizing the final end. First, as you know, whatever the method, the map with contours must be prepared. This secured, a tracing on cloth is laid upon the block of plaster and, with a pointed instrument, contours are transferred to the plaster before it has had time to dry thoroughly. You will begin then with the lowest contour and cut the plaster down to the desired level, this being determined upon by means of a sliding scale for measuring vertical elements. This done, the next contour is taken, and so on until there results a map like one made by building up with wood or other material. The details of relief have then to be worked in by carving down the shoulders as in any method of building up."

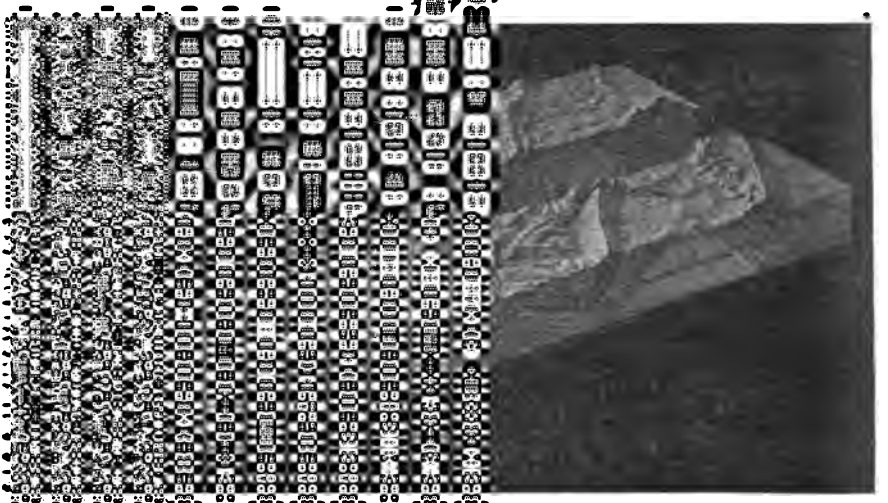
"The chief difficulty in this method is to be found in cutting up to the contours and securing a perfectly vertical edge, in order that the contour may stand properly when transferred to the lower level. The material carves easily, shrinks and swells so little, and may be revised with such great facility, that I prefer it to any other I have seen recommended. A little skill will enable one to work out the details."

We have found, that in using tracing-cloth upon damp plaster it is liable to change its shape and size, and thus destroy the accuracy of the result. In using tracing-cloth for transferring the map in the various methods above described, care should be used, since even on a dry surface, the pressure of the stylus will spread the drawing.

A method similar to that described by Professor King was practiced by Professor J. P. Lesley, at the University of Pennsylvania. Instead of using one solid block of plaster, a number of blocks of uniform size to cover the area of the map were taken, and a profile was drawn upon the sides of each block. They were then carved down to these lines, the spaces between the profiles being shaped by the eye.

One of the early models made in this country, that of the Morrison's Cove Region, Blair County, Pa., was constructed by Professor

of wood, three inches wide, ls. Fig. 4 is an illustration thod of construction, the two e model of Blair County, Pa., logical structure is shown in



ure in Blair County, Pa.

ing plaster blocks) of the cele-
igh County, Pa., and is now
Pennsylvania.

nry Brooks, Boston, described
ws :

es on waxed or oiled tracing-paper. ines when wetted. Paste the tracing ghth of an inch in thickness, and cut contour, leaving spaces now and then, ood would drop out. Fasten the ve- is flat. Fasten veneer by the edges, lines. Cut or have cut strips of thin of each contour line, and insert the veneer. They must be pressed down When all the contours are in place, , which will prevent the moisture of is coated, fill in the spaces between the brass show. Where spaces are

left the strips are cut with a slanting end long enough to span the space uncut, and the line of contour is thus unbroken. By this method nothing is left to the eye, and perfect accuracy is gained."

When a topographical map is engraved, a simple method of constructing a relief is to have a sufficient number of sheets printed directly upon cardboard of a proper thickness, to cut out the successive contours, and to fasten them together with a paste that will not swell the paper. Some of the European models exhibited at the Centennial Exhibition in 1876 were beautifully made in this way.

It is desirable that some other material than plaster be found to reproduce relief-maps in a rapid and inexpensive way. *Papier maché* is used by manufacturers of globes and other small reliefs, but it requires special apparatus. The writers have attempted a method in which blotting-paper is used, somewhat after the stereotyping process. After the plaster negative has been obtained, a large sheet of blotting-paper is thoroughly saturated with water and laid upon the mould. A brush with hard bristles is then used to beat the paper until it conforms to all the inequalities of the relief. It is then covered with paste made of:

Flour, ...	4 parts, made into paste.
Whiting, ...	2 "
Glue, ...	1 "
Water, a sufficient quantity.	
Mixed with heat.	

A second sheet of blotting-paper is then put on in the same manner, to be followed by another coat of paste, and a third sheet of paper, or more, if great stiffness is required. When the relief is very great, the paper may be broken in forcing it into the deeper parts of the mould, in which case small pieces of paper with paste can be used to fill up and repair the break. This must remain until dry before removal from the mould, when it may be given a thin coating of shellac-varnish and painted. The paper should be large enough to allow the edges to be turned up and glued at the corners, when the sheets will be stiff enough to frame without any further backing.

Mr. James B. Jordan, of London, England, who has made several large models, for various purposes, writes as follows:

"My method of modeling is by means of contours, which I believe is the only accurate one, but of course it involves the preparation of a carefully contoured map. My first large model was made in 1873, of the area of country drained for the

Sheffield water-supply. It embraced an area of 100 square miles. The horizontal scale was 880 feet to one inch, the vertical 500 feet to one inch (1 : 1.76). The surface was built up in contours of cardboard, at intervals of 25 feet, each card being $\frac{1}{20}$th of an inch thick, making the vertical scale a fair proportion for a moderately hilly country. The surface of the model was colored in distemper of a light buff-color, the reservoirs and rivers being shown in blue. The size of the model was 6 by 4 feet.

"In 1874 I made another important model for the use of a Parliamentary committee. This was also built up in contours of cardboard, at intervals of 10 feet. The horizontal scale was $\frac{1}{1584}$, the vertical scale $\frac{1}{2400}$ (1 : 4.4). The surface was colored geologically. The model was divided into two parts, having deep vertical edges, on each of which was colored a section of the strata, showing coal-seams, etc. For the purpose required this plan was found very useful.

"At every tenth contour a different colored cardboard was introduced to mark each 100 feet elevation. This is a great advantage and especially useful during the process of construction. The size of this model is about 9 by 5 feet.

"The cardboard I use for model work is made especially for the purpose and very hard. It is composed throughout of the best Whatman paper, cemented together with glue, instead of paste, as in the ordinary way.

"Another model now in process of construction is that of the British Islands, with a portion of the Continent of Europe and surrounding sea-bottom, in contours of cardboard at every 250 feet, commencing at the greatest depth of the ocean, 20,000 feet below sea-level. The cardboards number 25 to an inch. At each 1000 feet a light blue cardboard is inserted which gives a very pretty effect where the features are so undulating as in this model. It is being made for the Science and Art Department for educational purposes."

A notable geological model is that of the London area, based on the maps of the Geological Survey of Great Britain, and now in the Jermyn Street Museum, a description of which is given in a Guide to the Geology of London, by William Whitaker, F.G.S., published in the *Memoirs of the Geological Survey*, 1884. The horizontal scale is 6 inches to 1 mile, and the model represents an area 15 miles east and west, and 11 miles north and south.

It was found necessary to increase the vertical scale, as the greatest difference in level in the area represented—that between the Thames and the top of Hamstead Heath—would be little more than half an inch on the scale; consequently all the minor features would be lost. The vertical scale was therefore made 200 feet to one inch, an excess of 4.4 times over the horizontal. Owing to the large size of the model it was constructed in nine separate pieces and the lines of separation were chosen in view of the geological sections which could be represented. The blocks are therefore irregular in shape. The four corner blocks are fixed and the other five are movable by suitable mechanism to exhibit the sections.

We are indebted to Mr. J. B. Jordan, by whom the model was made, for a description of the method of construction:

"A firm foundation table was made with great care, to avoid any alteration in the form by shrinkage or warping of the timber. Nine deep frames, of the proper form, were made and fitted together on the foundation. Each frame top was covered with cotton cloth, and on these were mounted a set of the six-inch ordinance maps of the district.

"These maps were the datum plane on which the undulating surface of the model was raised. As there was no regular system of contour lines on the maps, then available, though they had elevations at numerous points, the system adopted was to drive measured pins into the points where the heights were marked, the pins, of which several thousand were used, being cut to exactly represent the elevation above the datum plane at the point through which they were driven. After driving the pins over a space representing a square mile, that space was covered with soft wax above the pins, and then the wax was worked down so as to show the bright end of each wire.

"Plaster casts were taken in suitable frames, and on completion of the nine areas they were fitted together, strengthened with sizing and a covering of muslin.

"To finish the surface a set of maps were mounted directly on the undulating surface, a task needing much care and patience, on account of the paper having to be thoroughly saturated and nearly reduced to pulp before being fixed on the surface of the model.

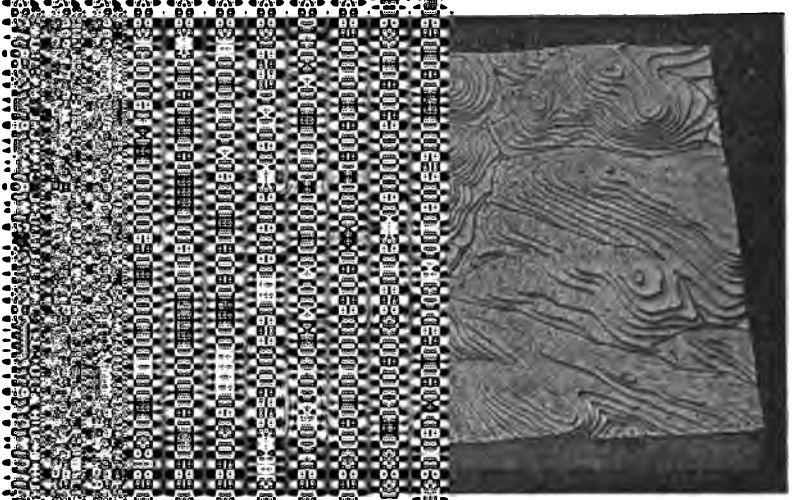
"Drawing paper was pasted on the sides of the blocks on which the geological sections were drawn, and the whole colored appropriately to represent the various formations."

The method preferred, however, by the writers, and the one that has given the most satisfactory results in their hands, as regards both accuracy and ease of manipulation, is that of cutting out the contours in wood of the proper thickness to represent the vertical interval selected, building up and finishing in wax.

Having a baseboard one inch thick, with one planed surface, and the exact dimensions of the intended relief, either paper, cardboard, strawboard, or wood of a proper thickness to represent the contour-curves, is used to build up the model. Having at hand transfer paper, stylus, hammer, wire-nails, glue and a jig-saw, you have all the material necessary to begin the work. Suppose the scale of your map to be $\frac{1}{19,200}$ of nature, or 1600 feet to the inch, and it is decided to build up in steps representing 50 feet vertically; then the material should be $\frac{1}{32}$ of an inch thick. Lay your map upon the baseboard and transfer to it the outline of the first 50-feet contour. Transfer the same outline to the material selected for the building up of your relief and also the outline of the 100-feet contour as a guide for the position of the next layer. With the jig-saw cut to the outline of the 50-feet contour, then fasten it to the baseboard with glue and nails in the position marked for it. Proceed in like manner to transfer the outline of the 100-feet contour and

150-feet layer, saw out the

elevation of the relief, always
one for cutting, the other for
yer above. Should the final
, be less than 50 feet, it may
proper height. When this is
ces $\frac{1}{12}$ of an inch in height,
responding in outline to the
o on to the highest elevation,
strated in Fig. 5.
should be laid down with the
layer, to counteract the con-



the South Mountains.

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be cut along certain section-
cture, these lines should be
d to each layer at the same
ailed down care must be taken
m these lines to allow the saw

The model is now ready for the waxing process. For this purpose the following formula is recommended:

Beeswax,	16 parts.
Venice turpentine	4 "
Corn starch,	8 "
Venetian red,	1 "
Sweet oil,	1 "

Melt the wax, add the Venice turpentine, and sift in the corn starch, stirring the mixture thoroughly, after which the Venetian red or other coloring matter may be added, together with the sweet oil. This preparation, poured into flat dishes, and allowed to cool, is used to fill up the angles or steps.

The skill of the topographer now comes into play in forming the finer details of the topography from the map, between the successive steps formed by the boards, it being assumed that the original is in contours 10 feet apart; and it is in this part of the work that the mere mechanic fails. The tools employed are of various rounded forms and sizes, to suit the intricate shapes of the finished model. They can be made of hard wood or bone. The wax should be of a consistency to work easily, which can be obtained by regulating the quantity of sweet oil in the above formula.

In some first attempts by one of the writers, a mixture of resin and tallow was used to cover the wood before applying the wax. In practice this was found unnecessary, the wax adhering to the wood without it. A little sweet oil used on the tools when giving the final touches, will be found advantageous.

If the above method of waxing is followed, the relief may be kept for years without danger of separation from the wood; or it may be finished for immediate use when time is an object. Usually, however, the wax relief is only used to cast a mould in plaster, from which any number of duplicates may be made. At this stage of the process help may be had from a plasterer, who will cast copies for a small sum, depending upon their size. When a modeler desires to make his own casts, it may be done in the following manner:

Enclose the relief in a wood frame, oiled or soaped to prevent the adhesion of the plaster, and place the whole on a flat surface to secure an even distribution of the liquid mass when poured into the mould. A sufficient quantity of water is then taken in a suitable vessel, into which plaster of Paris is carefully sifted through a fine-mesh sieve, rapidly stirring to prevent the formation of lumps and

air-bubbles in the mixture. When it arrives at the consistency of cream, it should be poured into the mould and allowed to stand until partially set. A straight-edge should then be drawn across the top of the frame to remove the surplus plaster, level with its upper edge. Large casts may be strengthened with iron rods or with wire netting inserted soon after the plaster is poured into the mould; and the weight of a thick cast may be reduced by hollowing out the plaster from the under side before it gets thoroughly hard. In a short time the plaster will be sufficiently hard to turn over, when the frame may be removed and the wax original separated from the cast. The writers prefer to dry the mould after retouching and to apply a coat of thin shellac varnish to the plaster negative before casting a positive. When time is an element, a positive may be made immediately by brushing the mould with a solution of soap-water before casting.

The finish of a model depends on the purpose for which it is to be used. For photographic prints or photo-mechanical reproduction, the surface features can be put directly upon the plaster in black, India-ink, or paint.

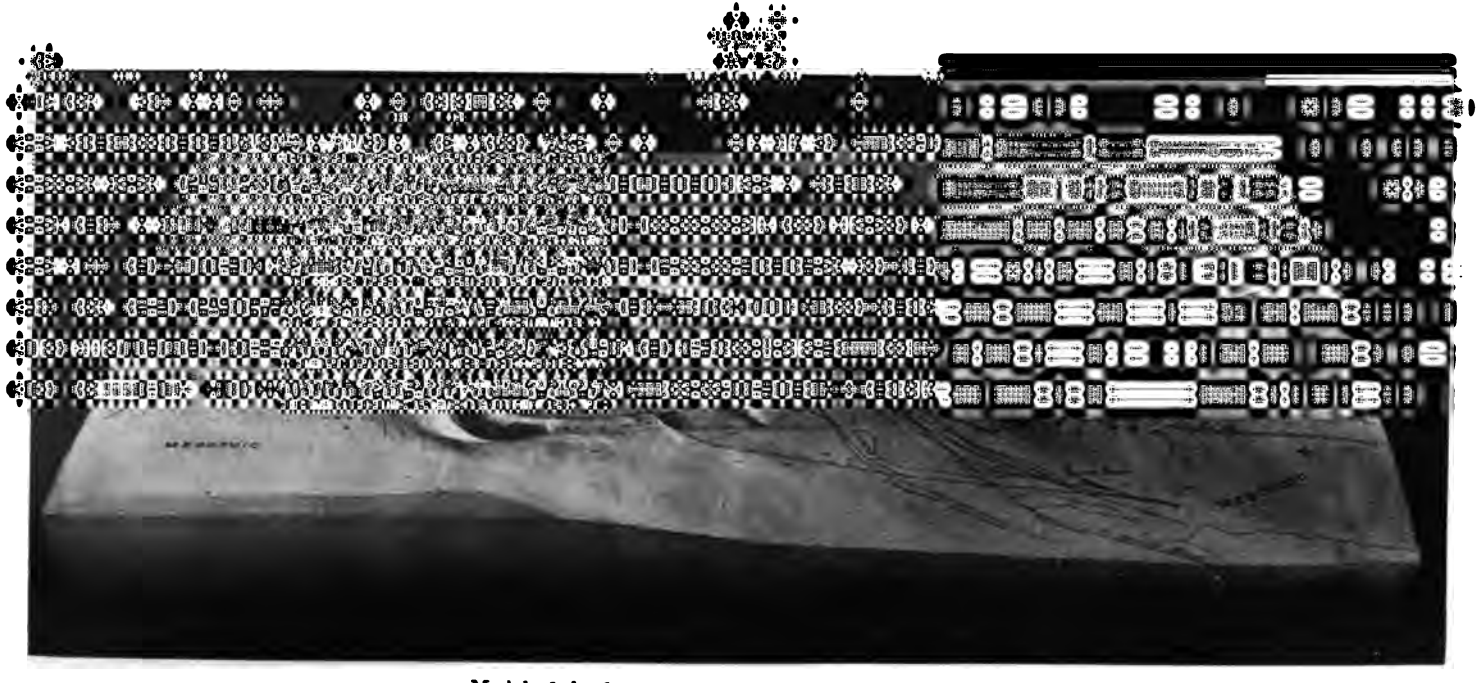
Fig. 6 is a relief of the western end of the Northern anthracite coal-field, prepared in India-ink for photo-mechanical printing by the Ives process. If it is desired to color the model geologically or otherwise, a thin coat of shellac-varnish is applied, and it may then be colored in oil.

The artistic map-maker will have no difficulty in giving character and finish to his cast, but unskilful drawing and poor lettering will ruin the appearance of otherwise good work.

The use of contour-curves to delineate underground topography originated with Mr. B. S. Lyman about 1865, and the first model based on his map was made to illustrate the complicated structure of the Cooper or Bennett coal-beds at the Hutchinson mine of the Plymouth Coal Company, Plymouth, Luzerne County, Pa.

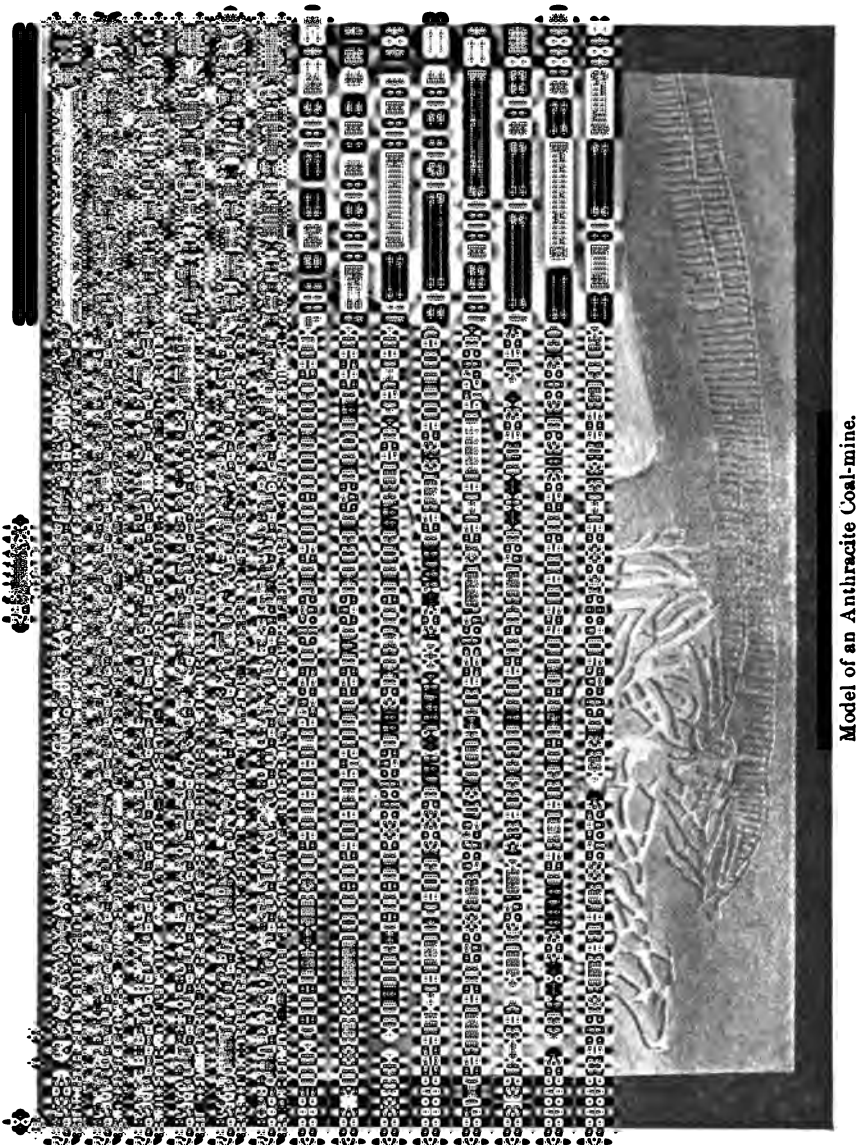
The method of modeling last described, is admirably adapted for the construction of underground relief; and a plate showing, on this plan, the workings in the middle member of the Mammoth coal-bed at Excelsior colliery, Northumberland County, Pa., is given in Fig. 7. Other models of underground topography constructed in the same way are those of the Panther Creek coal-basin (floor of the Mammoth coal-bed) between Tamaqua and Mauch Chunk, in Schuylkill and Carbon counties, illustrated in the Pennsylvania Geological Survey Report, AA. p. 242; also of the floor of the Mammoth coal-

Model prepared for Photo-mechanical Printing.



Model of the Cornwall Iron Mine as seen from the south.

Schuylkill County,
lisbury coal-basin,
ania.



Model of an Anthracite Coal-mine.

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Fig. 8, taken from

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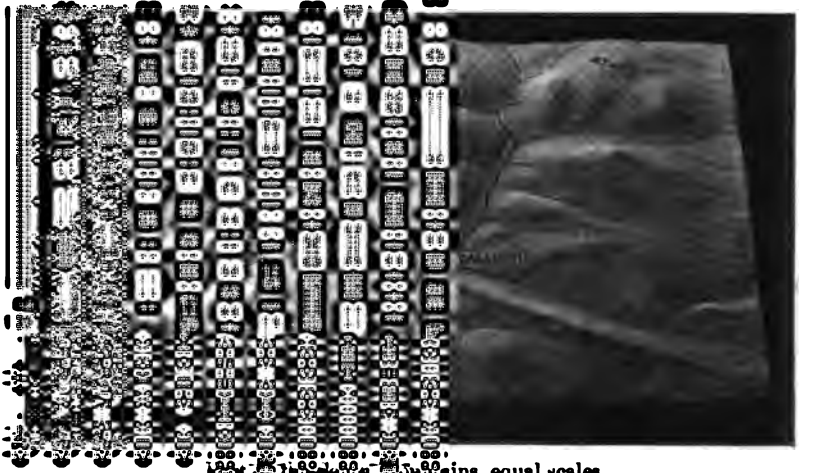

Mountains, equal scales.

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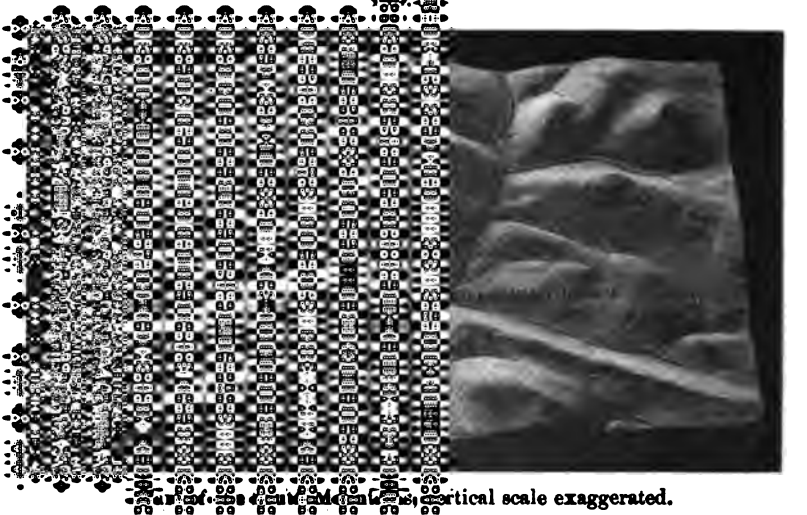
f maps, though the weight of



vertical scale exaggerated.

ghtly exaggerated scale. The
the distortion of the vertical

where possible, and believe it to be unnecessary in maps of a large scale, say of 100 to 1000 feet to one inch ($\frac{1}{12000}$ to $\frac{1}{12000}$), depending somewhat upon the nature of the topography. Fig. 9 exhibits a piece of South Mountain, Pa., topography, near Caledonia furnace, constructed on the true scale, vertical and horizontal alike, from map Fig. 1, where the greatest difference in elevation is 1000 feet.

Fig. 10 shows the same territory with the vertical scale increased as 2 to 1.

Mr. E. E. Howell, Rochester, N. Y., in a letter to us says,— "In nearly all the models I have made, the vertical scale and the horizontal scale are the same. In two maps covering a large area I doubled the vertical scale; but I believe, in every case where it is practicable, the two scales should be the same; and were I to-day re-modeling the two maps referred to, I would not exaggerate the vertical scales."

Prof. J. P. Lesley expresses the same opinion in vol. vii., page 58, of *Science*, where he says:

"I must personally thank you for your good words in behalf of non-exaggerated reliefs in your last issue, p. 24. I have had a long experience in this kind of work, and never found a case which required the vertical scale to be exaggerated. No relief of the surface is too delicate to escape the human eye when represented with sufficient skill and care in modeling. The demand for exaggeration in a relief comes from those who will not spend a sufficient amount of time and pains upon the intermediate contour curves, or from those who have not trained themselves in drawing from objects. The habit of exaggerating the relief excuses itself at first on the plea that common people cannot appreciate heights when true to nature, but the fact is that the difficulty is felt by the modeler himself, and, when the habit is once formed, it becomes incurable. If a relief-map be not true to nature, what is the good of it? Geologists have been forced to abandon exaggerated cross-sections; why should they permit relief-map makers to revive the discarded error, and put the representation of the whole in antagonism to the representation of the parts?

"About the year 1865 or 1866 I made a wooden model of one of our lower Silurian limestone valleys, with its bounding ridges, about 20 miles long. The model was about 18″ by 36″, in 12 bars of wood, each 18″ long by 3″ wide. On each side of each bar I painted the corresponding section of the valley, with its limonite horizons and faults. The model still exists. My purpose was first to get correct ideas of the country structure for my own work, and then to exhibit my conclusions to the Pennsylvania Railroad Company, who employed me. The reliefs in the valley were very low; but they were perfectly legible to the eye of a layman. What would have been the fate of my side-sections had I used an exaggerated vertical scale?

"In 1865 I made a model of the underground of the Plymouth anthracite mine, with its remarkable vertical fault, from levels which I took in the mine. What good would this have been had I used a different vertical scale?

"I have myself made models on several plans; the most satisfactory, but the most

laborious, being to draw a good many cross-sections on the same vertical and horizontal scale, along parallel lines, as nearly as possible at right angles to the general strike; then cut strips of wood, lead, zinc, or stiff paper (I have used all four) to represent the cross-sections, set these up in their places, fill in with wax or plaster, and finally tool the surface thus obtained. I prefer this method to the common one of jigging out the contour curves, and filling the terraces between them with slopes of wax. The latter method is easier and less costly; but it is sure to make the modeler slovenly in his geological representation, and it is a powerful seduction towards exaggeration of the vertical scale. Beginners and earnest scholars ought not to be allowed to use this method until they have been drilled to accuracy and to love the true natural aspect, by the compulsion of the method of cross-sections. I never see a false relief-map without indignation, and a touch of the contempt we feel for all anachronism."

Prof. F. H. King, River Falls, Wisconsin, who with Mrs. King has lately completed a model of the world on Mercator's projection, addresses the following letter to *Science*, vol. vii., page 120.

"There are various uses for topographical models, and that for which they are designed must necessarily govern their construction. While the technical geologist, in considering orographic questions, finds it undesirable to exaggerate the vertical scale of his cross-sections, such profiles would be absolutely useless in the actual construction of a railroad. It should be equally evident that the needs of school-children under sixteen years, and those of the field geologist, are not necessarily met by identical appliances. The construction of suitable topographical models for use in the common schools is educationally of the utmost importance, and, now that the matter has been referred to, I hope it may receive the consideration it demands. Almost every great physiographic and commercial problem requires the pupil to see his locality and State in its vertical relations to other States and countries; and how best to enable him to do this, is not solved by Professor Lesley's dictum.

"What we need to-day for educational purposes, as I see it, is an accurate topographic model of every State in the Union, constructed in such proportions as will enable the pupils, in their respective schools, to use it as a working-plan for the making of a larger model of their State. This map should not be isolated. The pupil must see it in its horizontal and vertical relationships to other States. Now, to meet these demands, a relief-map of the United States is required, in which both the horizontal and vertical elements for each State may be measured with sufficient accuracy and facility by the pupil. Such a model must be portable, very strong, and *extremely cheap*. I emphasize the last, because, unless they are cheap, the schools needing them most cannot have them. Now, a model of the United States might be constructed, as Professor Lesley suggests, but it would be useless for topographic purposes if made of any portable size. Our own map has the horizontal scale sixty-five miles to the inch, and it is certainly as large as can be conveniently handled in the average school-room. But taking the Grand Cañon district as an example of what might be done with both scales alike, using Mr. Dutton's profile, extending from the Markagunt plateau southward across the Grand Cañon, for data, we should have the following profile:

1. Markagunt plateau,	10,568	feet	above sea-level,	or	.0295 inch.
2. North bank of Parunuweap,	4,659	"	below (1) "	"	.0138 "
3. Depth of bed of stream,	1,250	"	below (2) "	"	.0036 "
4. Height of Vermilion Cliffs,	1,818	"	above (2) "	"	.0053 "
5. Foot of Vermilion Cliffs,	1,363	"	below (4) "	"	.0040 "
6. Brink of Permian terrace,	1,022	"	above (5) "	"	.0030 "
7. Foot of cliff,	568	"	below (6) "	"	.0016 "
8. Brink of second terrace,	1,022	"	above (7) "	"	.0030 "
9. Foot of second terrace,	1,931	"	below (8) "	"	.0057 "
10. Brink of Grand Cañon,	113	"	above (9) "	"	.0004 "
11. Bed of Colorado,	1,363	"	below (10) "	"	.0040 "

"These figures are a sufficient proof of the impracticability of making a model of any large section of country without exaggerating the vertical scale, to say nothing of cheaply reproducing it with any degree of accuracy. Our map, constructed with the horizontal scale 5000 feet to the inch, that is, the same as the vertical, would be about 16 rods long and 9 rods wide. Were it constructed with the vertical scale the same as the horizontal, Mount Whitney would be but .044 of an inch high; Mount Washington, .018 of an inch; and the highest point in Wisconsin, .0053 of an inch. Our model has attached to it one of the summits of the White Mountains, both scales alike, covering a rectangle 9 by 5 inches, and shows in itself just what the effect of exaggeration is. For my part, when I think of a mountain valley represented on the model, I think of it as 65 times wider than it is in the model; and I believe that pupils, if properly taught, will do so."

In the catalogue of the Royal Geographical Society's exhibit of appliances used in geographical education, London, 1885, we find 11 reliefs constructed on the same horizontal and vertical scales; on 8 the information is not given; 9 were "exaggerated many times," or "considerably;" and over 20 were exaggerated in their vertical scales from 2 to 72 times.

Probably one of the largest relief-maps or models ever made was of the earth, 60 feet in diameter, covering 10,000 feet of surface and exhibited by Mr. Wyld in Leicester Square, London, in 1851. The horizontal scale was 10 miles, the vertical 1 mile to 1 inch ($\frac{1}{633,600}$ to $\frac{1}{63,600}$). The view was taken from the interior, the delineation being on the concave instead of the convex surface. There was no writing upon the model; the sea was colored blue, and the land of as natural a tint as possible.

Lieutenant H. A. Reed, in his work on topographical drawing, recommends the following exaggeration as affording a pleasing relief: "For a map scale 6 inches to 1 mile; if mountainous, $\frac{1}{3}$; if only hilly, $\frac{1}{2}$; if gently undulating, $\frac{2}{3}$. For smaller scales, except for very rugged tracts, the exaggeration should be correspondingly increased. For a tract consisting wholly of mountains, no exaggeration is necessary."

Relief-maps are invaluable to the instructor in geographical and geological teaching, and this fact was early recognized at the University of Pennsylvania, where, in 1873, Prof. J. P. Lesley had prepared by and under the direction of one of the writers, assisted by the students in the department of geology and mining, a number of models based on actual surveys, to illustrate special features of geological structure. A list of these is here given.

1. The Schuylkill Water Gap in the Blue Mountain, at Port Clinton, Pa., size 38 by 32 inches, scale $\frac{1}{4800}$.

2. The Delaware Water Gap in the Blue Mountain, size 27 by 25 inches, scale $\frac{1}{9600}$.

3. The Lehigh Gap in the Blue Mountain, size 25 by 21 inches, scale $\frac{1}{4800}$.

4. The Black Log Gap and environs of Orbisonia, Huntingdon County, Pa., size 25 by 22 inches, scale $\frac{1}{9600}$.

5. Part of the Northern Anthracite coal-field, from Shickshinny to Scranton, Luzerne and Lackawanna Counties, Pa., based on a topographical map by Mr. R. P. Rothwell, size 180 by 40 inches, area 80 square miles, scale $\frac{1}{12,000}$.

6. Logan's Gap, in Jacks Mountain, Pa., size 18 by 18 inches, scale $\frac{1}{4800}$.

7. The underground workings at the Oak Hill colliery, Drifts Nos. 1 and 2, Scranton, Lackawanna County, Pa., size 12 by 15 inches, scale $\frac{1}{1200}$.

8. The Ironton Railroad Company's brown hematite mine, Lehigh County, Pa., size 50 by 18 inches, scale $\frac{1}{600}$.

9. The Washoe Mining District, Nevada, based on United States Geological Survey maps.

10. The White Pine mining district, Nevada, based on United States Geological Survey maps.

11. The Iron Mountain, Mo., based on the State Survey map.

12. The New Jersey zinc-mining district, Franklin Furnace, Sussex County, based on the State Survey map.

13. The property of the Cambria Iron Company, adjacent to their iron-works, Cambria County, Pa., scale $\frac{1}{7200}$.

14. A part of Big Sewell Mountain, Fayette County, West Va., showing the number and position of the coal and limestone beds on the lands of the Longdale Coal and Iron Company, from private surveys.

The following is a list of models constructed for and based upon the work of the Geological Survey of Pennsylvania:

1. A part of Blair County, published in report T, scale $\frac{1}{19,200}$.

2. The Seven Mountains Regions, Union, Mifflin and Snyder Counties, Pa., scale $\frac{1}{63,360}$.

3. The same, cut into 14 sections to show the geological structure.

4. Part of the South Mountains, Adams and Franklin Counties, Pa., from sheets A1, A2, B1 and B2, published in D5 atlas, scale $\frac{1}{19,200}$.

5. Part of the Mahanoy and Shenandoah coal-basin, showing the floor of the Mammoth coal-bed, scale $\frac{1}{9600}$.

6. The Panther Creek coal-basin from Tamaqua to Mauch Chunk, showing the floor of the Mammoth coal-bed, scale $\frac{1}{9600}$.

7. Part of the Bald Eagle mountain and Tyrone Gap, Huntingdon County, Pa., showing faulted strata, horizontal scale $\frac{1}{19,200}$, vertical $\frac{1}{9600}$. Published in report T3, page 369.

8. Rocky Ridge and East Broad Top coal-basin, Huntingdon County, Pa., horizontal scale $\frac{1}{19,200}$, vertical $\frac{1}{9600}$. Published in report T3, page 285.

9. The Loyalsock coal-basin, Sullivan County, Pa., scale $\frac{1}{19,200}$.

10. The Jones magnetite mine, Berks County, Pa., scale $\frac{1}{600}$.

11. The Cornwall magnetite mine, Lebanon County, Pa., scale $\frac{1}{9600}$. Annual Report, 1885, p. 490.

12. The Cornwall Mining Company's property, Lebanon County, Pa., scale $\frac{1}{19,200}$.

13. The ancient channel of the Allegheny River at Parker, Clarion County, Pa., scale $\frac{1}{126,720}$.

The following is a list of models made for private use:

1. Morrison's Cove and part of the Allegheny Mountains, Pa., scale $\frac{1}{96,000}$.

2. The Gabel, Jones and Gabel Magnetite Mine, Boyertown, Berks County, Pa., used in suit for trespass, scale $\frac{1}{600}$.

3. The scene of the Pickering Valley railroad accident, near Phœnixville, Chester County, Pa.

4. The Cumberland Valley, from the Susquehanna to the Potomac, showing the country adjacent to the line of the Cumberland Valley Railroad, based on geological survey maps. For illustration of this model, see *Trans.*, xiv., 446.

5. The Pottsville Gap, from data furnished by the Pennsylvania Railroad Company.

6. The southwestern end of the Northern Anthracite coal-field, Luzerne County, Pa., based on Geological Survey maps, N.C.F., mine sheets Nos. 1 and 2, horizontal scale $\frac{1}{9600}$, vertical $\frac{1}{4800}$.

A series of models to illustrate geographical and geological teaching have been designed by Prof. N. S. Shaler and executed by Prof. W. M. Davis, of Harvard University, assisted by Mr. T. W. Harris.

These models are not intended simply to represent known localities, but to illustrate types of structure and forms of the simpler kind, and are planned from beginning to end with regard to their use in teaching. With this in view, they have in moderate amount the same kind of exaggeration so commonly employed and permitted in black-board diagrams; the fact of this being in three dimensions instead of two is not sufficient reason for divesting them of the advantages of emphasized and idealized types, and reducing them to mere copies of actual places. The generalization that they present to the student is as intentional as it is useful. The danger of error in such models may be lessened, if not substantially avoided, in two ways: first, by basing every model on examples personally familiar to the designer, and this has been done as far as possible; secondly, by supplementing the series with models of actual places constructed from accurate surveys or with photographs. For this purpose photographs have been employed, exact models being rarely made, and when made, entirely too expensive for school use.

Besides the generalized expression of the relation of geographic form to geologic structure that these models embody, they are prepared in sequences, so as to illustrate geographic evolution, or the successive changes of form that appear as a given structure is further and further denuded. Thus a single sequence begins with a plateau reduced to a rugged country by the increase in the number and breadth of its valleys expressing maturity; then the same again at a later stage when the relief has been greatly diminished and the previously rugged surface has been toned down to a gently rolling country, as in old age; this surface is then shown buried in a smooth glacial sheet, in illustration of the kind of accidents that a region is subject to in the course of its long life; the fifth model shows the ice-sheet nearly melted away, leaving the old valley more or less obstructed with drift, and the streams held back in a lake, with its outlet turned aside from the old drainage-channel; and finally, the lake thus formed disappears, its water-level being marked by terraces, as the outlet cuts down a gorge such as frequently occurs in our northern States.

The student thus gains a glimpse of comparative geography as well as of structural geology, and finds how the present form of one

region may represent the early stage of another. Although the models are on a small scale and many details of form are necessarily omitted, they have been shown by actual use to serve a good purpose in presenting facts that are conceived with difficulty from maps and diagrams alone.

The collection consists of 25 models 5 by 7 inches in size, and is accompanied by an equal number of photographs of actual structures, to illustrate more fully the subjects of the models.

The importance of relief maps for educational purposes was shown at the exhibition of appliances used in geographical education, held by the Royal Geographical Society of Great Britain, in December, 1885, when a large number of such maps were exhibited by the various countries of Europe.

It is desirable that a cheaper method of duplicating models be found; and experiments are being made by the writers towards this end. Plaster of Paris is for many reasons a good material; but its weight and its liability to damage in transportation render it unsuited. Paper seems to be best adapted to meet the requirements of the case, and we hope at a future time to give the results of our endeavors in this direction.

LA PLATA DEL LIBANO MINES, DEPARTMENT OF TOLIMA, REPUBLIC OF COLOMBIA, SOUTH AMERICA.

BY WILLARD IDE PIERCE, NEW YORK CITY.

(Boston Meeting, February, 1888.)

THE Department (formerly called State) of Tolima, which is beginning to attract considerable attention in this country on account of its mineral wealth, comprises an area of 18,415 square miles, lying between latitude 1° 34′ 25″ and 5° 44′ 12″ N. and between longitude 74° 25′ 45″ and 76° 34′ 25″ W. from Greenwich. It is bounded on the north by the Department of Antioquia, on the east by the Departments of Cundinamarca and Cauca, and on the south and west by the Department of Cauca.

It is the most recently created of the States of Colombia, having been established by decree of April 12th, 1861, and being composed of the old provinces of Mariquita and Neiva, and of the districts Melgar, Cármen, Cunday and Santa Rosa, of Tequendama.

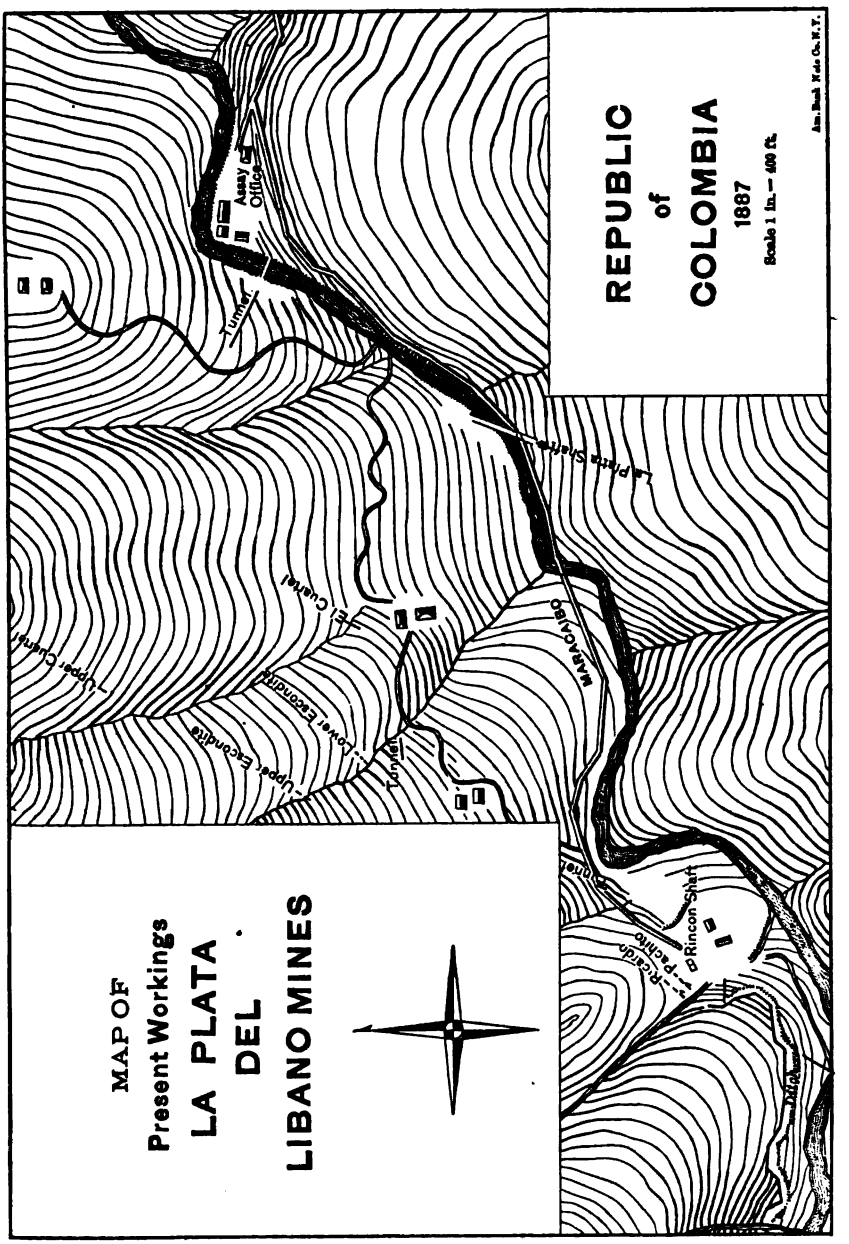
The eastern and central cordilleras of the Andes pass from north to south through the Department, enclosing almost all of it between their summits. The name of the Department comes from the snow-covered mountain, Nevado del Tolima, which lies in the Central Cordillera and rises to a height of 18,420 feet above the level of the sea.

The Magdalena river, which rises near the southern extremity of the Department, is of the utmost importance to its development. Steamers run from the Atlantic coast to the City of Honda, a distance of 550 miles; and from that part smaller steamers go as far south as Purificacion. The river separates the eastern and central cordilleras, and the plains of the Magdalena lie between the river and the central cordillera. Two seasons only are known, the wet and the dry, divided about as follows: from the latter part of December until the first part of March, dry; then wet until the middle of June; then dry until the middle of September; then wet until the middle of December. The wet and dry seasons are also known by the names of winter and summer, but there is not much difference in the temperature, the dry season being a little the warmer. From the founding of the cities of Mariquita and Ibagué, in 1551, mining may be said to have been continuously carried on up to the present time; and more activity is now shown than at any previous recent period.

Through the courtesy of the Compañia de Minas del Libano of Bogatá, I am permitted to make some remarks about the mines of this company. The mines are situated in the District of Libano, in the central cordillera of the Andes, about 3500 feet above sea-level, and are about 14 leagues southwest from Honda and 8 leagues west of Ambalema, both cities on the Magdalena river. The plains of the Magdalena are about 4 leagues distant, while the town of Libano lies about 3 leagues north from the mines. The property owned by the company comprises 1773 acres (718 hectares), well-wooded, and bearing, amongst other varieties, diablo fuerte, dinde, diomate, guyacan-baho, guyacan-tomé, and other hard woods well fitted for mining-timber, charcoal, etc.

The known veins consist of two separate and distinct series, one of which comprises the La Plata vein, and the other the La Esperanza group, a series having the same general direction, dip and characteristics.

La Plata vein, the only one of the first series exposed thus far, is a fissure-vein with walls of quartzose mica-schist. It was first worked


MAP OF
Present Workings
LA PLATA
DEL
LIBANO MINES
REPUBLIC
of
COLOMBIA
1887
Scale 1 in. = 400 ft.
Assay Office
Tunnel
La Plata Shaft
El Cuartel
Upper Cuartel
Lower Escondite
Upper Escondite
Tunnel
MARACAIBO
Rincon Shaft
Ricardo
Pachito

by means of an open cut, but later a shaft was sunk which is 65 feet deep. The shaft is 13 x 9 feet and divided into two compartments. It is timbered and lagged at the ends, stulls only being used between the sides of the shaft, as the walls are good and at present do not need other timbering. There are 2 drifts running southwest on the vein from the shaft, the upper 9 feet below the top of the shaft, or on a level with the road, and 170 feet long, and the lower drift 50 feet below the upper and 47 feet long. The veiu has been traced on the other side of the creek but no other work has been done on it. The vein is vertical, and has a direction of N. 20° E.

The ore consists of quartz carrying gold and silver combined with iron pyrites and galena and some zinc blende, as well as a small amount of copper pyrites. The name of the vein is misleading, as there is a large proportion of gold combined with the silver; but this name was given on account of the results of the assays from the out-crop, which showed but little gold. The distance between the walls is 10 feet and the quartz vein varies from 3 inches to 5.5 feet.

La Esperanza group consists of several fissure-veins which were but slightly opened at the time of my visit there; but from what could be seen I consider them to be parallel veins. While the strike of each as found at the different openings varies somewhat, still, when further opened they will probably be found to have a general direction of N. 40° E.

Prospecting is difficult on account of the luxuriant vegetable growth; and almost all the veins have been found by following the different water-courses. It is not probable that the whole number of veins on the property is known, but a sufficient number has, however, been found to demonstrate that the property is a valuable one. There are being driven one cross-cut and two tunnels to cut these veins at different depths; and these will determine more accurately their directions and relations with one another.

At the Rincon there is what appears to be, as near as can be determined from the work done, a vein with a width between the walls of 60 feet, and presenting a banded structure. The walls are of quartzose mica-schist. Three of the bands of the vein are of quartz and occupy positions, one each at the foot and hanging walls and the other about half-way between; the two intermediate bands are quartzose rock of no value. The Rincon shaft, which is sunk on the foot-wall band, is down 53 feet, well timbered with timber 1 foot square and lagged with 4 inch plank. It consists of two compartments each 8 x 9 feet in the clear. For the present but one of

these compartments will be continued to a greater depth. At a depth of 52 feet a cross-cut is being run from the east compartment to cut both the Pachito and Ricardo, as the central and hanging-wall bands are called. The Pachito and Ricardo are both opened by short tunnels and show good, strong veins at the face of the tunnels. They are each 18 inches wide with a direction of N. 40° E. and dip 60° to the northwest. These bands present much the same general appearance and the ore there, as in the other veins of this group, consists of quartz with free gold and small amounts of iron pyrites and galena.

The Upper Escondite presents a vein of quartz from 20 to 24 inches wide with a direction of N. 30° E. and dip 38° to the northwest. The Lower Escondite is a similar vein with a direction of N. 45° E. and dip 33° to the northwest. A tunnel is being driven to cut both the Upper and Lower Escondite and will cut the latter at a depth of 50 feet.

The Upper Cuartel is 3 feet wide with a direction of N. 30° E. and dip 65° to the northwest, and the Lower Cuartel is 1 foot wide with a direction of N. 25° E. and dip 52° to the northwest. Metallic silver was found in a sample taken from the latter but all the veins of this group contain free-milling gold-ore with a small amount of sulphurets. With perhaps one exception, it is probable that all these veins are separate and distinct, and that their prolongation has not yet been opened. The exception referred to is the Lower Cuartel (marked "El Cuartel" on the map), which may belong to the series at the Rincon. Care must be taken, in tracing the prolongation of these veins on the map, to make allowance for the different elevations at which the various veins have been uncovered, as well as for the different dips as found, which as has already been mentioned will probably have to follow one general direction as the veins are further opened.

The Upper Escondite is 185.5 feet above the level of the Rincon shaft, while the Lower Escondite is 110 feet above the same shaft, and the Upper Cuartel and Lower Cuartel are respectively 252.5 feet and 100 feet above the level of the shaft at the Rincon.

The power for pumping and hoisting at the Rincon and La Plata shafts, as well as sufficient for at least a 10-stamp mill, can be obtained direct from the Quebrada La Plata, which runs near the two shafts and mill-site, by means of over-shot or turbine water-wheels, there being sufficient fall to allow of the same water being used at the several different places. For pumping and hoisting at the

Rincon shaft the water will be conveyed in a ditch 1800 feet long. Most of the ditch is already dug and ready for the water, with the exception of the necessary flumes across the ravines, and where the nature of the ground is such that boxes must be placed on the ditch to carry the water. The end of the ditch at the shaft is 39 feet above the top of the shaft; and from the shaft to the mill-site, marked "Maracaibo" on the map, there is a fall of 42.5 feet. From the mill-site to the top of La Plata shaft there is a fall of 31 feet, and from the top of La Plata shaft to the level of the creek at the assay office there is a fall of 41 feet. The Quebrada carries sufficient water, even in the dry season, for all these purposes; and the same water after being used at one place, can be carried by ditch to the next, and so on. If at any time greater power should be needed, a ditch can be dug tapping La Mediadera Quebrada, about 3 miles off, which will furnish a large supply.

The situation is a most convenient one, permitting not only the continuous use of the water, but such a location of the mill-site that a tunnel, run from the level of the breaker-floor of the mill, will cut all the veins (with the exception of the veins at present worked at La Plata and Rincon shafts) at a depth of 125 feet for the lowest opening and at a proportionate depth for the others.

The ore is easily worked, breaking readily, and can be mined at small expense. Wages are low, the price of labor (September, 1887) varying from 16 to 27 cents* per day for boys and women, to 86 cents for blacksmiths. Miners' wages vary from 33 to 65 cents per day. None of these prices include board.

The climate and temperature at the mines are all that could be asked for, it being a very healthy situation and the temperature during August, which is considered one of the warmest months of the year there, varying from 64° F. in the early morning to 82° F. in the shade in the middle of the day (18° to 28° C.). Machinery, etc., needed for the mines from the coast, could be brought by steamer to Ambalema, and from there across the plains by wagons or mules; from the plains to the mines there is a good, easy mule-road, along which *cargas* can be readily brought.

* All values are in U. S. gold.

PHOTOGRAPHING THE INTERIOR OF A COAL-MINE.

BY FRED. P. DEWEY, NATIONAL MUSEUM, WASHINGTON, D. C.

(Duluth Meeting, July, 1887.)

In preparing material for the exhibit of the National Museum at the New Orleans Exposition in 1884, it was decided to attempt to photograph the interior of a coal-mine, in order to get a strictly truthful representation of a thick coal-seam, and to show incidentally the methods of mining. While we have many drawings, both mechanical and artistic, illustrating the subject, they are not satisfactory for these purposes, simply because they are drawings.

Aside from this, much interest attaches to pictures of inaccessible places, and the enlarged copies (30 inches by 40 inches) of the photographs we succeeded in obtaining, have attracted a large amount of attention from the visitors to the Museum.

The Kohinoor colliery, Shenandoah City, Pa., was selected as most suitable for the attempt, principally on account of the fine exposure of the Mammoth bed, and the comparatively level character of the workings, in that colliery.

This is one of the many anthracite-mines of the Philadelphia and Reading Coal and Iron Company. A detailed section of the fifty-foot or Mammoth bed in this colliery is as follows:

Feet.	Inches.		
1	0	Clod.	Top Split.
7	0	Coal.	
1	0	Slate.	
8	0	Coal.	
1	6	Slate divider.*	
6	0	Coal.	Bottom Split.
0	6	Slate.	
2	6	Coal.	
0	5	Slate.	
1	8	Coal.	
1	2	Charcoal.	
5	0	Coal.	
1	0	Bone and Coal.	
0	6	Bone and Slate.	
3	0	Coal.	
0	10	Bone and coal.	
1	2	Coal.	
0	2	Bone.	
4	4	Coal.	
0	1	Bone.	
0	8	Coal.	

* Varies from 1 foot to 25 feet in thickness.

On the 10th of August, 1884, an attempt was made to get the light for a picture by burning one-half ounce of magnesium wire. Owing to a variety of causes, this was a total failure as to producing a picture, but showed conclusively that it would be very difficult, if not impossible, with the facilities at hand, to succeed by the use of magnesium.

Mr. S. B. Whiting, General Manager of the Philadelphia and Reading Coal and Iron Company, became very much interested in the undertaking, and after the failure with magnesium, afforded every facility possible for having a thorough trial made with the electric light, in order to settle the question whether the interior of a coal-mine could be photographed or not.

The dynamos of the Company at Reading were too large for use in the mine, but, upon Mr. Whiting's suggestion, Mr. R. I. Kear, Secretary of the Arnoux Electric Light Company, kindly offered to supply all the electric machinery and material that might be required, and personally took charge of setting up and running the lights. The Museum is much indebted to Mr. Kear for this assistance.

The engine, dynamo, lamps and all the necessary apparatus were first set up and tested at the Pottsville shops. Everything being found satisfactory, the plant was then taken to the Kohinoor colliery.

It was originally intended to establish the dynamo at the air-compressor house, but, upon arrival on the ground, it was found that the wire provided was not long enough to reach so far, and it was, therefore, decided to take the whole plant 400 feet below ground. The work of transferring the plant from the railroad-car on the surface and setting it up in running order below ground was accomplished in six hours.

The electric plant consisted of a five-light Arnoux dynamo, driven by a 9-inch by 12-inch horizontal engine, with balanced piston-valves. These machines were securely bolted in line on 6-inch by 8-inch timbers resting on blocks, and the whole was securely braced against the roof.

The engine was supplied with two heavy fly-wheels, five feet in diameter, and the pulley on the dynamo, 8 inches in diameter, was connected directly with one of these by a 5-inch leather belt. To obtain the proper speed of the dynamo, 1200 revolutions per minute, the engine had to run at 160 revolutions.

The engine was driven by compressed air from a 2-inch branch

of the main supply-pipe of the mine, which ran four hoisting-engines, two fans, and two pumps. Occasionally, on account of the heavy demands of the hoisting-engines, the pressure was considerably reduced, so that the speed of the dynamo was sometimes only two-thirds what it should have been. Owing to the heavy fly-wheels of the engine, the changes in speed were quite gradual, and no violent changes were observed in the intensity of the lights.

The plant was established in an enlargement of the main gangway at the head of the long slant gangway, and Breast No. 39, about 1300 feet away, was selected for photographing. About 4000 feet of wire were required to complete the circuit.

Each of the five lights used was nominally of 2000 candle-power, but on this occasion they gave a light of only 1600 candle-power each, the loss being due mainly to induction, on account of the excessive moisture in the atmosphere of the mine.

To support the lights, two pieces of scantling were fastened together at the top and spread at the bottom, the lamps hanging between. These holders could easily be changed about to secure the proper illumination, and could be leaned against any convenient support, as a mine-timber, an empty car, or even the solid coal itself, to steady them. Rough reflectors were improvised from pieces of tin plate for some of the lights.

Breast No. 39 was selected for photographing because it shows forty-two and one-half feet thickness of coal, and because the process of taking out the roof and supporting pillars of coal, or "robbing the mine," was going on there. Six months previously, shots had been fired in the top-coal, throwing down immense quantities of coal, which had been nearly all removed. The breast is under the western edge of Shenandoah City, about 500 feet vertically below the surface.

Everything being in readiness, the dynamo was started late in the evening of August 28th, and a scene of surpassing beauty was revealed in Breast 39. For the first time could be seen at one glance its whole forty-two and one-half feet of alternating layers of coal, "bony," and slate, which could readily be distinguished by the most inexperienced through the difference in luster, while the large pile of coal that had been blasted from the roof, containing many lumps of several tons weight, was an interesting sight.

All the miners within sight quickly gathered to see this most unusual spectacle, and as the news of the brilliant illumination spread through the mine, a large number left their work, and came to swell

the crowd. Many old miners were attracted to the spot, and were, if possible, more surprised and interested in the sight than the strangers present. Although most of them had spent the greater part of their lives in coal-mines, yet they had never before seen more than a few square feet of the coal at any one time.

The scene also had its dark side, since the bright light clearly revealed the great dangers from the fall of coal and slate to which the miners are constantly exposed. The bonds which held large masses, often of many tons weight, to the roof and sides, were clearly seen to be very slender and frail, and looked as if they might give way at any time. This clear revelation of dangers, before only dimly appreciated, must have an important bearing upon the question of the introduction of the electric light into coal-mines for practical mining purposes.

One negative was taken at this time; but the atmosphere was very thick with smoke and dust, and it was not quite successful.

On the following morning the novelty of the undertaking had somewhat worn off, and, with the experience already gained, the work of taking the negatives proceeded much more rapidly and satisfactorily. Seven exposures were made, and five of the resulting negatives were good. Four of the views have been copied by the Levytype process, directly from the negatives, without any retouching, to illustrate this paper.

The following observations will facilitate the examination of the views:

First Exposure. Fig. 1 shows on the right in the foreground the corner of a pillar of coal, which exhibits in great perfection the banded structure of the formation. Near the center are two miners in characteristic positions, at work removing the pile of coal that has been thrown down by the blast in robbing the mine. At the left in the background, is the face of the breast, which has been squeezed or crushed by pressure, as is shown by the confused appearance of the strata.

The camera was placed in the breast near the entrance on the left side, three lights were placed in the room, on the left of the camera, and two behind it in the gangway. Time of exposure, ten minutes. The Museum number of this view is 59,007.

Third Exposure. Fig. 2 has the face of the breast for a background, showing especially the lower benches of the formation, and the slate partings. A man on the left (whose complexion, by the way, is given with evident accuracy) is breaking down the coal with

a pick; one in the center is loading the lump-coal on a mine-car with a shovel, and one on the right is drilling a hole for a blast, with the old-fashioned hand-drill or "jumper." The scraper for removing the drill-dust from the hole and the needle for the introduction of the cartridge lean against the face of the breast in front of the middle man. Time of exposure, ten minutes. Museum number, 59,006.

Fourth Exposure. Fig. 3 shows nearly the same background as Fig. 2. A miner is drilling a hole in the seven-foot bench for a blast, with a modern patent drill and by the light of the safety-lamp, various tools being grouped about him. Time of exposure, fifteen minutes. Museum number, 59,008.

Fifth Exposure. Fig. 4 shows the entrance to the breast. At the left in the background is a pillar of coal, left to support the roof. In front of this is the mine-railway. In the center, in the background, is the entrance to the breast, with the roof (at this place the five-foot bench), supported by timber props. On the right, in the background, is another pillar of coal; in front of this is a pile of refuse or "gob;" and in front of this, again, are some worn-out timber props. A man, standing in the entrance, furnishes a measure of the height of the gangway. The camera was placed at the face of the breast. Time of exposure, thirty minutes; but during this exposure the lights were dimmed several times by the heavy demands of the hoisting-engines upon the supply of compressed air. Museum number, 59,009.

The photographic work was performed by Mr. George M. Bretz, of Pottsville, Pa., who kindly supplies the following information: The lens used in some of the exposures was a Steinheil No. 6, and in the others a Ross view-lens. A medium-sized diaphragm was used in every case. The plates were Carbutt's special (JC) keystone dry plates, 8 inches by 10 inches. Pyrogallic acid and carbonate of potash developers were used, and the plates were developed as soon as taken, in order that inspection of the results might guide the operator in subsequent exposures.

For a dark room, a flat car, which had served in many ways in setting up the apparatus, was simply pushed around the corner into the usual darkness of the mine, which was illuminated by an ordinary glass lantern with a piece of manilla paper tied around it, giving a reddish glare.

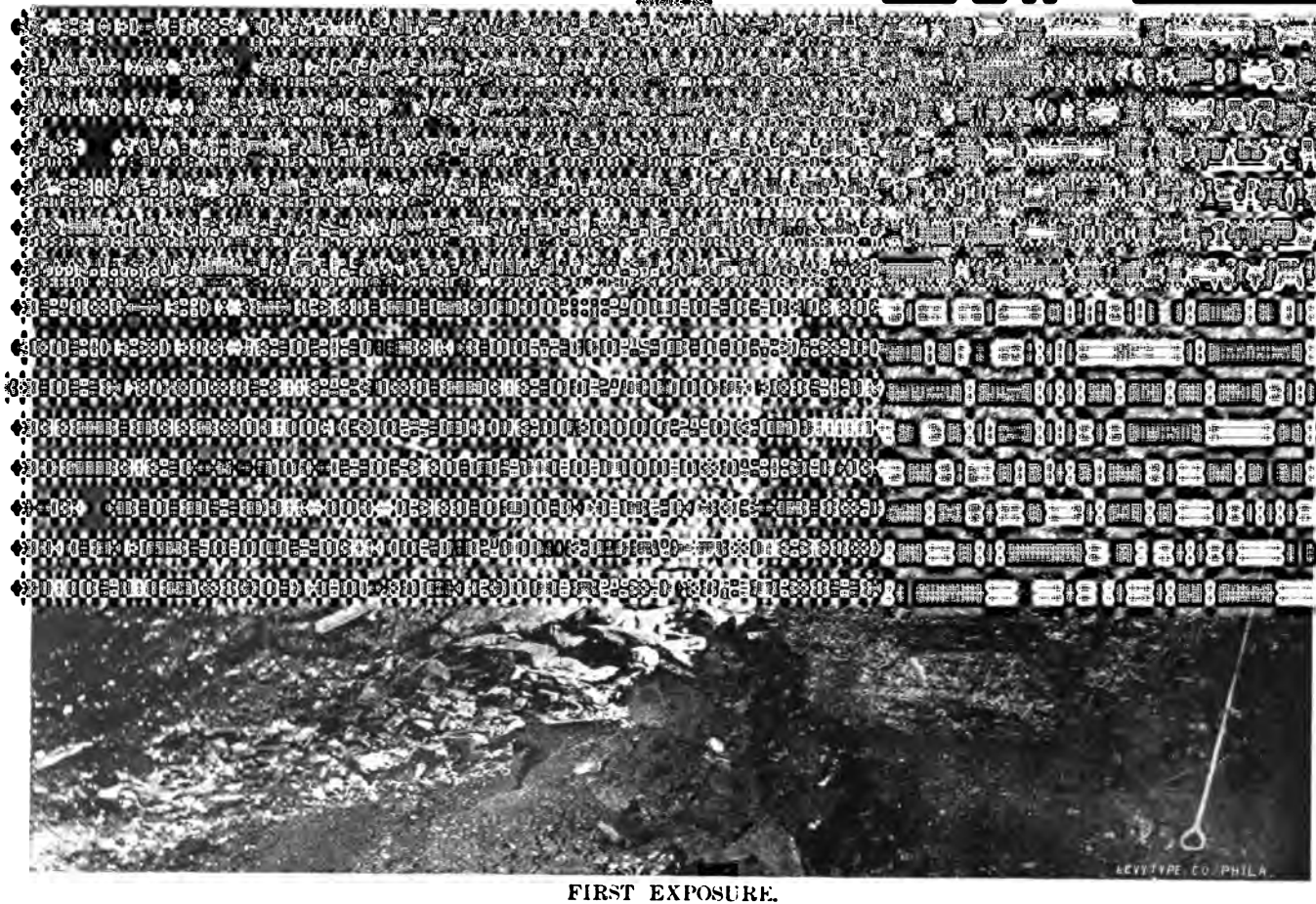
The difficulties are not many, but are hard to overcome. They include smoke from lamps and powder burned at blasting, and coal-

dust, which not only thickened the atmosphere, but deposited black particles on the plates, and in the water and reagents; the dripping of water upon the apparatus; the condensation of water upon the plates and lens; and the strong reflection from the brilliant and glossy surface of the coal when the light struck it at the proper angle. This last difficulty was so unexpectedly great as to give much trouble, the reflection being so strong in some cases as to obscure all details, and give simply a blotch of white.

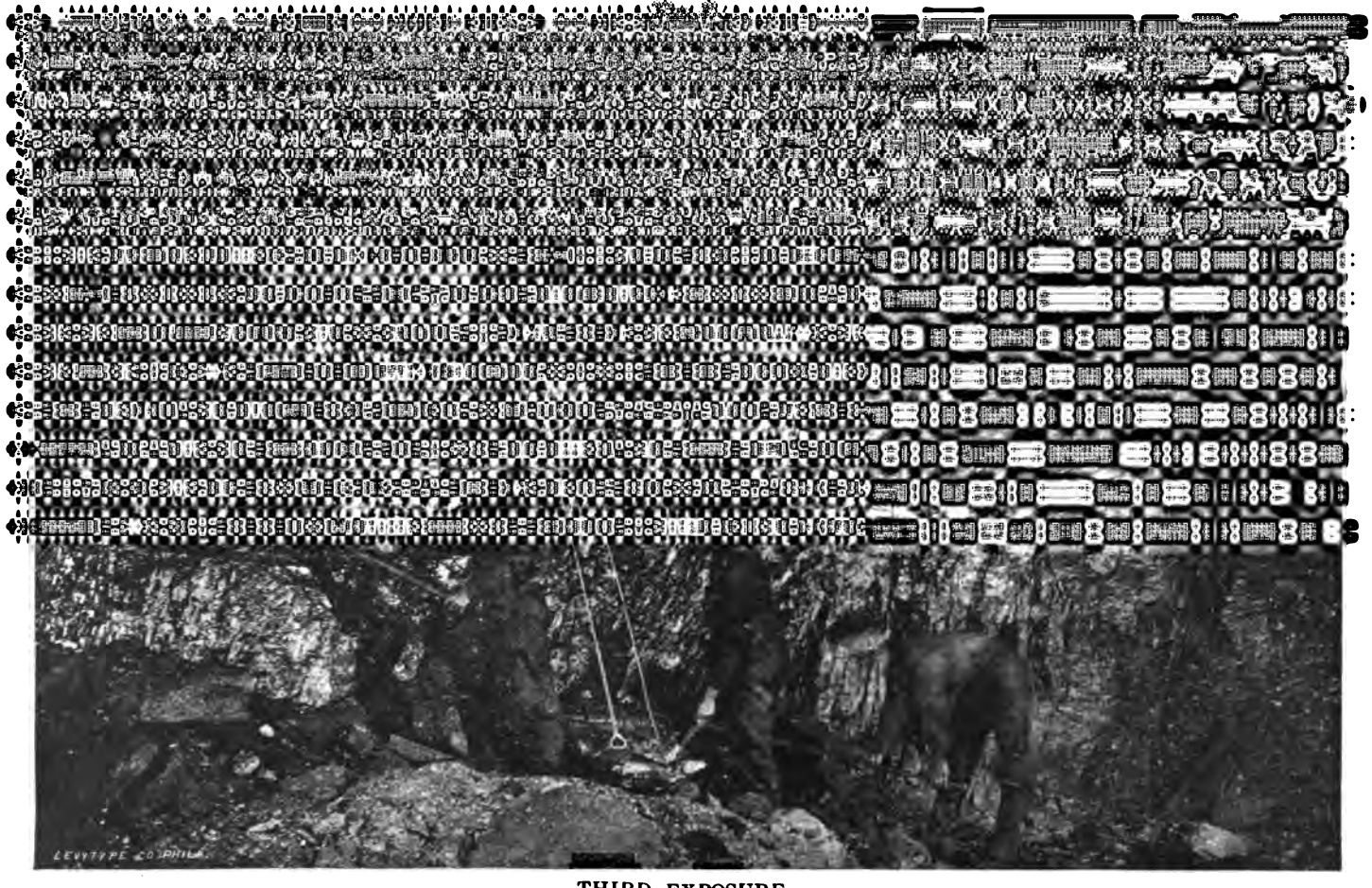
No tests of the power required to run the lights were taken at this time; but, on the occasion of the visit of the Institute to the Indian Ridge Colliery, on September 6th, 1884, when the members had the unusual pleasure of seeing a whitewashed coal-mine, a complete set of indicator cards was taken. It had been originally planned to retain the plant that we had used in photographing at Kohinoor, and invite the Institute there; but, owing to the better facilities for taking care of a large number of people, it was afterwards decided to remove the plant to Indian Ridge Colliery. The same dynamo was used, but a steam-engine furnished the power, and seven lights were used instead of five. The plant was 310 feet below ground. Repeated trials have shown that .875 horse-power per light are required under ordinary circumstances above-ground, but in this case 1.26 horse-power per light were required, being an increase of 44 per cent.

A single exposure for an hour and a half, using the smallest diaphragm, was made by Mr. Bretz at that time in the Indian Ridge Colliery; but he reports that the negative was defective. The breast was illuminated for the benefit of the visitors, and not for photographing; it was impossible to place the camera so that the light would not fall directly upon the lens, and this produced broad rays of light upon the negative.

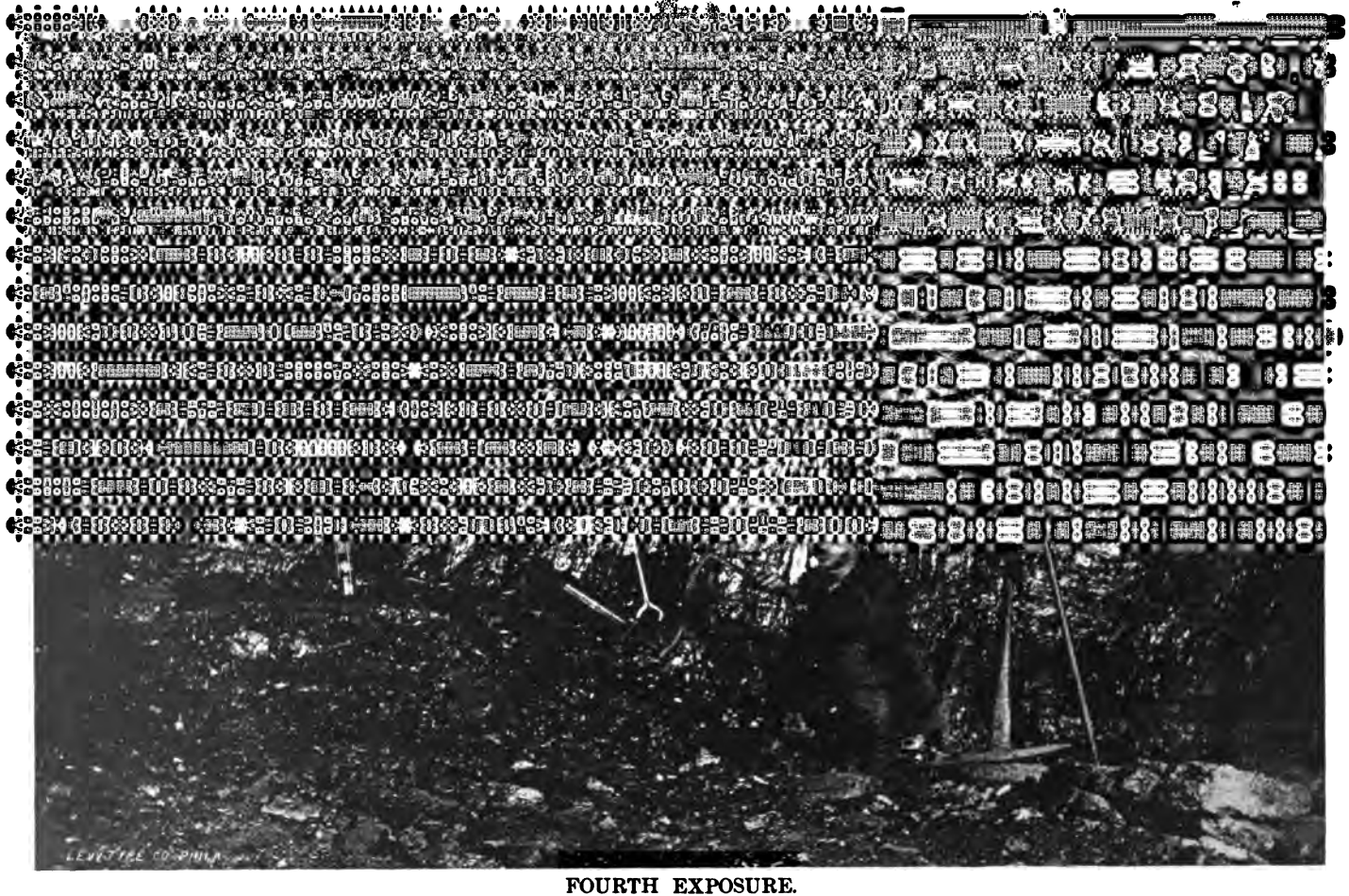
The execution of this interesting and important undertaking was under the immediate supervision of the late J. T. Brown, of the Museum staff, and its success is due to his untiring energy and perseverance. For many privileges and kindly interest in this work, the Museum is greatly indebted to Mr. S. B. Whiting, General Manager, Philadelphia and Reading Coal and Iron Company, and to Mr. E. F. C. Davis, Superintendent of the Pottsville shops of the Company.



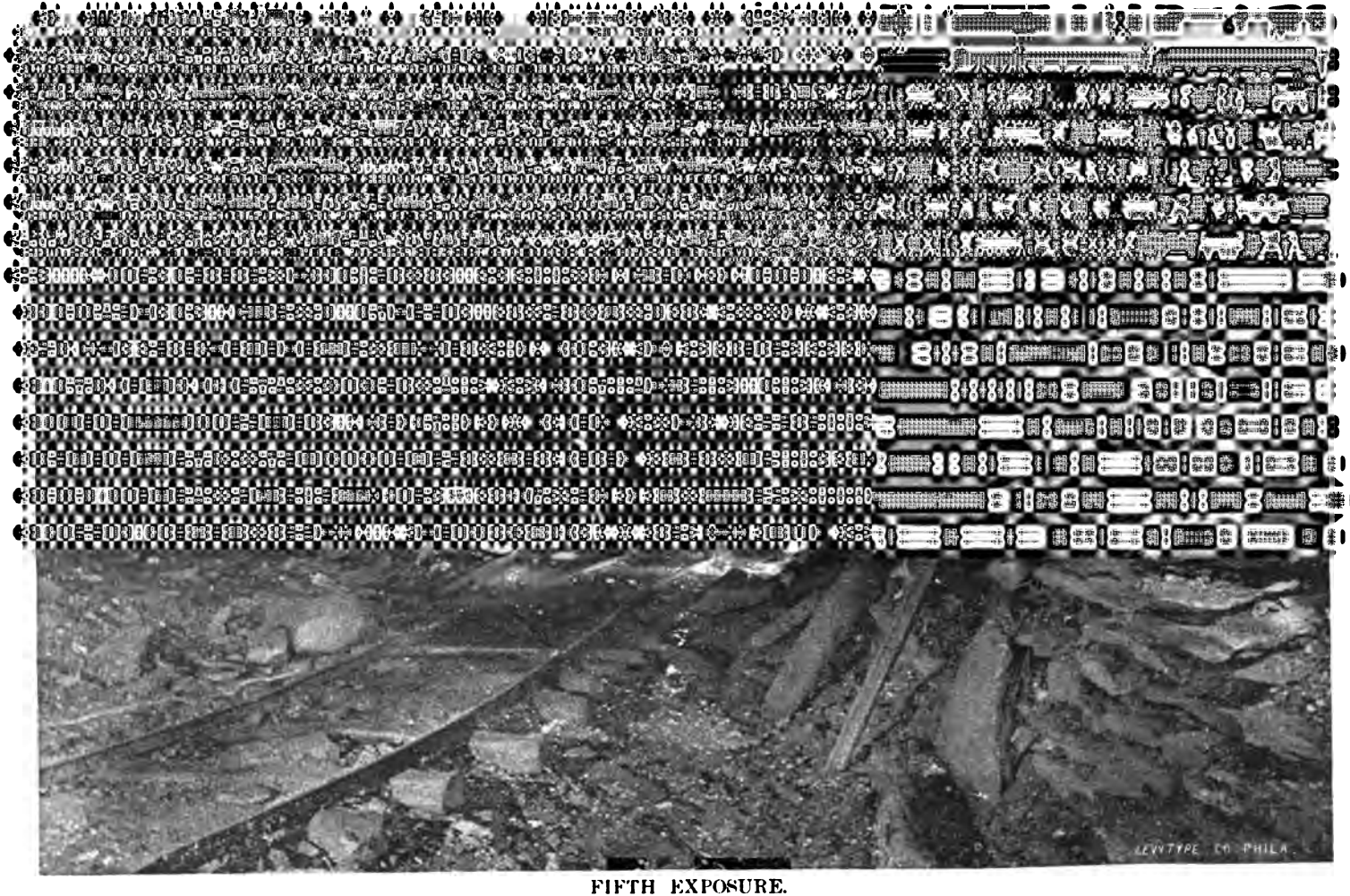
FIRST EXPOSURE.



THIRD EXPOSURE.



FOURTH EXPOSURE.



FIFTH EXPOSURE.

A GLOSSARY OF FURNACE-TERMS IN ENGLISH, FRENCH AND GERMAN.

BY THOMAS EGLESTON, PH.D., NEW YORK CITY.

(Boston Meeting, February, 1888.)

THE uncertainty of finding the exact equivalents for technical expressions in different languages has led me to think that a glossary of furnace-terms would be useful to members of the profession. I was led, some years ago, to prepare a small one, which has been found so very useful, that I have arranged one for presentation to the Institute, for four different types of furnaces, in the hope that it may prove of use to those who are in the habit of consulting foreign technical journals. I was further induced to make the attempt in the hope that it might tend to make the terminology less complicated than it now is, and also to make it more uniform over larger districts. The idea of referring to a drawing which expresses exactly what is meant, resulted from the necessity of presenting such subjects to students, and will, it is hoped, prove much more useful than the mere dictionary arrangement in alphabetical order.

The drawing of the blast-furnace was prepared for me by Mr. J. P. Witherow, of Pittsburgh; the drawing of the open-hearth furnace by Mr. S. T. Wellman, of Cleveland, and the reverberatory furnace by Mr. John Fritz, of Bethlehem. The bloomary-furnace is taken from my own paper* on Bloomaries, but, in order to have a complete furnace, I have introduced into the drawing of those used at Saranac the guard-plate used at Ironville. The interest which has been taken in the preparation of this paper by those gentlemen who have contributed the drawings, has been supplemented by gentlemen in Europe who have taken the trouble to read the proof. In France it has been read by Mr. Lodin, of the School of Mines, and by Mr. Chaper, of Paris, and in Germany by Dr. Wedding, of Berlin, and Mr. Luhrman, of Osnabrück.

The preparation of the paper has been much more difficult than was anticipated, and its value has been very much enhanced by Dr. Raymond's kind assistance and coöperation; and to him, as well as to all the other gentlemen, I am very much indebted. It was at first proposed to add to the paper the names of all the tools used in these various furnaces. It was found, however, that this would make such

* *Trans.*, viii., 515.

a series of synonyms in each language, the exact equivalent of which in other languages it would be so difficult to get, that the plan was abandoned.

It is hoped that the glossary, as it is, may prove to be useful, and that it will, to some extent, render more intelligible the terms used, and also that it will promote the unification of the terms over large areas. It will be a very great gain, if the publication of this paper may induce others to prepare similar glossaries in other departments of the profession, as such publication would, undoubtedly, tend to the simplification of terms used, and to the complete elimination of many of them.

I. THE FORGE.

(FIGS. 1, 2, 3, AND 4.)

ENGLISH.	FRENCH.	GERMAN.
Bloomary fire, Forge fire. Low hearth.	Bas Foyer.	Das Rennfeuer. Das Luppenfrischfeuer. Das Frischfeuer.
1. Water-cooled bottom-plate (Figs. 1 and 3).	1. La taque de fond à courant d'eau.	1. Der wassergekühlte Frischboden.
2. Tuyere-plate (Figs. 1, 2, and 3).	2. La taque de varme.	2. Der Formzacken.
3. Tuyere-plate repair-piece (Fig. 3).	3. La pièce rapportée de la varme.	3. Das Ausbesserungsstück des Formzackens.
4. Merrit-plate (Figs. 1 and 3).	4. La partie supérieure de la taque de varme.	4. Die Formzackenfeueeplatte.
5. Fore-spar plate (Figs. 1 and 2).	5. La plaque de contrevent.	5. Der Windzacken; Der Gichtzacken.
6. Fore-spar plate repair-piece (Fig. 2).	6. La pièce raportée du contrevent.	6. Das Ausbesserungsstück des Windzackens.
7. Skew-plate (Fig. 1).	7. La partie supérieure de la plaque de contrevent.	7. Die Windzackenfeuerplatte.
8. Fore-plate (Figs. 2 and 3).	8. La plaque de travail.	8. Der Arbeitszacken; die Arbeitsplatte; die Deckplatte.
9. Cinder-plate (Fig. 3).	9. La plaque de laiterol.	9. Der Brustzacken; der Vorderzacken.
10. Back-plate (upper part of the haire-plate) (Fig. 3).	10. La partie supérieure de la plaque de rustine.	10. Der Hinterzacken.
11. Haire-plate (Figs. 2 and 3).	11. La plaque de rustine.	11. Der Aschenzacken.
12. Foundation - plates (Figs. 1 and 3).	12. Les plaques de fondation.	12. Die Deckplatten.
13. Guard-plate (Fig. 1).	13. La plaque de protection contre les cendres.	13. Die Schutzplatte.

English.	French.	German.
14. Mantelpiece (Fig. 3).	14. Le sommier en fonte.	14. Der Mantel.
15. Bosh-trough (Figs. 1 and 2).	15. La bache à eau.	15. Der Wasserkasten; der Kühltrog.
16. Wind-trunk (Fig. 1).	16. Le conduit du vent.	16. Die Windleitung.
17. Blast-valve (Fig. 1).	17. Le registre du vent.	17. Das Windventil; die Drosselklappe.
18. Bed-pipe (Figs. 1, 3, and 4).	18. Le tuyeau principal.	18. Das Hauptwindrohr.
19. Hot-air pipes (bow-pipes) (Figs. 1, 3, and 4.	19. Les tuyaux à air chaud.	19. Die Winderhitzungsröhren.
20. Water-supply for merrit-plate (Fig. 1).	20. Le conduit d'eau froide pour la varme.	20. Die Kaltwasserleitung für die Formzackenfeuerplatte.
21. Water-discharge from merrit-plate (Fig. 1).	21. Le tuyau de décharge pour la varme.	21. Das Abflussrohr von der Formzackenfeuerplatte.
22. Water-supply for the tuyere (Fig. 1).	22. Le conduit d'eau froide pour la tuyère.	22. Die Kaltwasserleitung für die Form.
23. Water-discharge from tuyere (Fig. 1).	23. Le tuyau de décharge pour la tuyère.	23. Das Abflussrohr von der Form.
24. Water-supply for the tuyere-plate (Fig. 1).	24. Le conduit d'eau pour la plaque des tuyères.	24. Die Kaltwasserleitung für den Formzacken.
25. Water-discharge from the tuyere-plate (Fig. 1).	25. Le tuyau de décharge pour la plaque des tuyères.	25. Das Abflussrohr von dem Formzacken.
26. Water-supply for the bottom-plate (Fig. 1).	26. Le conduit d'eau pour la plaque de fond.	26. Die Kaltwasserleitung für die Bodenplatte.
27. Water-discharge from the bottom-plate (Fig. 1).	27. Le tuyau de décharge pour la plaque de fond.	27. Das Abflussrohr von der Bodenplatte.
28. Tuyere-saddle (not shown).	28. L'Assise de la tuyère.	28. Der Formsattel.
29. Stack-bonnet (Fig. 1).	29. Le couronnement de la cheminée.	29. Der Schornsteinhut.
30. Star-bucks or corner-irons (Fig. 1).	30. La cornière.	30. Das L-Eisen; das Winkeleisen.
31. Jamb-plates (Fig. 4).	31. L'armature de la base de la cheminée.	31. Der Kaminbeschlag.

II. THE REVERBERATORY.

(Figs. 5, 6, and 7.)

English.	French.	German.
Reverberatory Furnace.	Le four à réverbère.	Der Flammofen.
1. Ash-pit (Fig. 5).	1. Le cendrier.	1. Der Aschenfall.
2. Ash-pit doors (Fig. 5).	2. Les portes du cendrier.	2. Die Thür des Aschenfalls.

ENGLISH.	FRENCH.	GERMAN.
3. Grate-bar bearers (Figs. 5 and 7).	3. Les supports des barreaux de la grille.	3. Die Rostträger.
4. Grate-bars (Fig. 5).	4. Les barreaux de la grille.	4. Die Roststäbe; die Rostschwellen.
5. Fire-place or chamber (Figs. 5 and 7).	5. La chauffe.	5. Der Feuerraum; der Heitzraum.
6. Stock-hole door-frame (Fig. 5).	6. Le cadre du foyer.	6. Der Rahmen der Feuerthür.
7. Stock-hole door (Fig. 5).	7. La porte du foyer.	7. Die Feuerthür.
8. Slide for stock-hole door (Fig. 5).	8. La coulisse de la porte du foyer.	8. Die Gleitfläche der Feuerthür.
9. Door counter-weight (Fig. 5).	9. Le contrepoids de la porte.	9. Das Gegengewicht der Thür.
10. Counter-weight support (Fig. 5).	10. Le support du contrepoids.	10. Der Träger des Gegengewichts.
11. Combustion-chamber (Fig. 5).	11. La chambre à combustion.	11. Der Verbrennungsraum.
12. Fire-bridge (Fig. 5).	12. L'autel ou le pont de chauffe.	12. Die Feuerbrücke.
13. Fire-bridge wall (Fig. 5).	13. Le mur de l'autel de chauffe.	13. Die Mauer der Feuerbrücke.
14. Air-chill (Fig. 5).	14. La bâche à courrant d'air froid.	14. Die Luftkühlung.
15. Water-chill (Figs. 5 and 7).	15. La bâche à courrant d'eau.	15. Die Wasserkühlung.
16. Working- or charging-door (Fig. 5).	16. La porte de travail.	16. Die Arbeitsthür; Die Einsatzthür.
17. Working-hole (Fig. 5).	17. Le trou de travail.	17. Die Arbeitsöffnung; Die Kratzenthür.
18. Frame of working-door (Fig. 5).	18. Le cadre de la porte de travail.	18. Der Rahmen der Arbeitsthür.
19. Water-cooled sill (Fig. 5).	19. Le seuil de porte à courrant d'eau.	19. Die Schwelle mit Wasserkühlung.
20. Laboratory; kitchen; pot; hearth (Fig. 5).	20. Le laboratoire.	20. Der Schmelzraum.
21. Fore-plate (Fig. 5).	21. La plaque d' avant.	21. Die Schaffplatte; die Arbeitsschwelle.
22. Cinder-tap (Fig. 5).	22. Le trou de coulée des scories.	22. Das Schlackenloch; der Schlackenstich.
23. Bottom-plate (Fig. 5).	23. La plaque de sole.	23. Die Herdplatte.
24. Beam for bottom-plate (Fig. 5).	24. Le sommier portant la plaque de sole.	24. Der Bodenplattenträger.
25. Hearth (Figs. 5 and 7).	25. La sole.	25. Die Sohle.
26. Balance for charging-door (Fig. 5).	26. Le contrepoids de la porte de chargement.	26. Das Gegengewicht der Arbeitsthür.
27. Flue- or neck-bridge (Fig. 5).	27. L'autel du rampant; la plaquele petit autel.	27. Die Fuchsbrücke.
28. Flue or neck (Fig. 5).	28. Le rampant.	28. Der Fuchs.
29. Roof or arch of the furnace (Fig. 5).	29. La voute.	29. Das Gewölbe; das Herdgewölbe.

English.	French.	German.
30. Flue-roof (Fig. 5).	30. La voute du rampant.	30. Die Fuchsdecke; das Fuchsgewölbe.
31. Chimney or up-take (Figs. 5 and 6).	31. La cheminée.	31. Der Schornstein; die Esse.
32. Outside plates (Figs. 5 and 7).	32. Les plaques de revêtment.	32. Der Eisenmantel; die Armatur.
33. Bearers; anchor-bars (Fig. 5).	33. Les montants.	33. Die Stehanker; die Ankerschienen.
34. Ties (Fig. 5).	34. Les tirants.	34. Die Zuganker; die Zugstangen der Esse.
35. Cinder-tap (Fig. 6).	35. Le trou de coulée des scories.	35. Das Schlackenloch der Esse.

III. THE OPEN-HEARTH.

(Figs. 8, 9, 10, 11, 12, 13 and 14.)

	English.	French.	German.
	Open-hearth Furnace.	Le four Martin-Siemens.	Der Martin-Ofen. Der Siemens-Martin-Ofen
Front. / Le devant. / Die Vorderseite.	A. Fore-plate (Figs. 9, 12 and 14).	A. La plaque de devant.	A. Die Vorderplatte.
	B. Arch-plate (Figs. 9 and 14).	B. La plaque du cintre.	B. Die Gewölbeplatte.
	C. Cheeks, right and left (Figs. 12 and 14).	C. Les plaques de droite et de gauche.	C. Die Wangen, rechts und links.
	D. Bracket, carrying fore-plate (Figs. 9 and 14).	D. La console de la plaque antérieur.	D. Der Träger der Vorderplatte.
	E. Backstays (Figs. 9, 12 and 14).	E. Les supports des cadres.	E. Die Stützen des Rahmens.
Bottom. / Le fond. / Der Boden.	F. Side-plate (Fig. 12).	F. Les plaques de côté.	F. Die Seitenplatten.
	G. End-plates (Fig. 9).	G. Les plaques de bout.	G. Die Endplatten.
	H. Bottom-plates (Figs. 8 and 9).	H. Les plaques de sole.	H. Die Bodenplatten.
	I. Tapping-hole pl'e (Fig. 9).	I. Le trou de coulée.	I. Die Platte des Abstichs.
	J. Columns carrying bed (Fig. 8).	J. Les supports de la sole.	J. Die Stützen der Bodenplatten.
	K. Columns carrying I-beams (Figs. 8 and 9).	K. Les supports des poutres soutenant le sol.	K. Die Balkenträger des Bodens.
	L. Washers for channel-iron (Fig. 8).	L. Les rondelles des tirants des côtés.	L. Die Ankerscheiben des Rinneisens.

ENGLISH.	FRENCH.	GERMAN.
M. Washers for rails (Figs. 9 and 12).	M. Les rondelles pour les rails.	M. Die Ankerscheiben der Schienen.
N. Columns for trussed rails (Fig. 12).	N. Les colonnes pour les rails armés.	N. Die Säulen für die armirten Schienen.
O. Washers for trussed rails (Fig. 12).	O. Les rondelles pour les rails armés.	O. Die Ankerscheiben für die armirten Schienen.
P. Reversing-valve (Figs. 9 and 14).	P. Le clapet de renversement.	P. Das Gas- und Luftventil.
Q. Mushroom-valve (Fig. 9).	Q. La tête du clapet de renversement.	Q. Der Teller des Luftventils.
R. Butterfly-valve (Figs. 9 and 14).	R. Le clapet à papillon.	R. Die Drosselklappe; die Wechselklappe.
S. Bonnets (Fig. 14).	S. Les chapeaux.	S. Die Kappen.
T. Chimney-damper (Fig. 9).	T. Le clapet de la cheminée.	T. Das Ventil des Schornsteins; der Rauchschieber.
U. Gas-ports (Fig. 9).	U. Les entrées de gaz.	U. Die Gaswege; die Gaskanäle.
V. Air-ports (Fig. 9).	V. Les entrées d'air.	V. Die Luftwege; die Luftkanäle.
W. Regenerative gas-chambers (Figs. 8 and 13).	W. Les régénérateurs à gas.	W. Die Gasregeneratoren; die Gaswärmespeicher.
X. Regenerative air-chambers Figs. 8 and 13).	X. Les régénérateurs à air.	X. Die Luftregeneratoren; die Luftwärmespeicher.
Y. Checker work in chambers (Fig. 8).	Y. Les empilages des briques.	Y. Das Gitterwerk oder Würfelwerk in den Wärmespeichern; in den Kammerfüllungen.
Z. Pockets (Fig. 8, 10 and 13).	Z. Les poches à poussieres.	Z. Die Schlackentaschen; die Schlackenfänge
1. Flues from chambers to furnace (Fig. 8).	1. Les carneaux à gas chauds.	1. Die Steigkanäle für Luft und Gase.
2. Bottom (Figs. 8 and 9).	2. La sole.	2. Der Boden.
3. Bed; hearth (Figs. 8, 9 and 12).	3. Le laboratoire.	3. Der Herd.
4. Roof; arch (Figs. 8, 9 and 10).	4. La voute.	4. Das Gewölbe.
5. Spout (Fig. 12).	5. Le trou coulée.	5. Der Stich.
6. Gas-flues from producer (Fig. 11).	6. Les conduites de gaz du générateur.	6. Die Gasleitung von dem Generator.
7. Chimney-flue (Figs. 9, 11 and 14).	7. Le carneau allant à la cheminée.	7. Der Schornsteinfuchs.
8. Gas-flues, from valve to chamber (Figs. 9, 11 and 14).	8. Les conduites de gaz du clapet vers le four.	8. Die Gasleitung vom Ventil nach dem Herde.
9. Air-flues, from valve to chamber (Figs. 11 and 14).	9. Les conduites d'air du clapet vers le four.	9. Die Luftleitung vom Ventil nach dem Herde.

IV. THE BLAST-FURNACE.

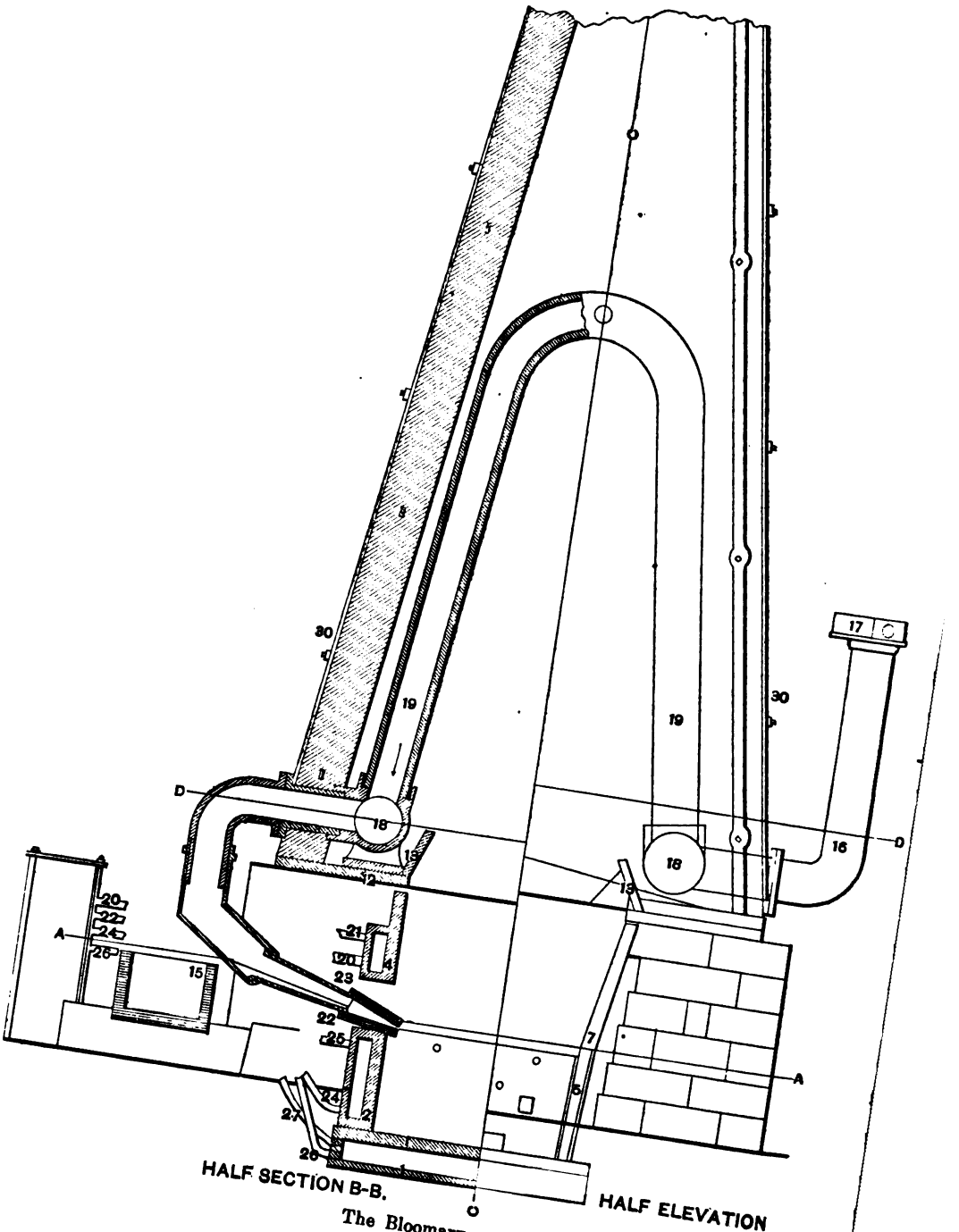
(FIG. 15.)

ENGLISH.	FRENCH.	GERMAN.
1. Foundation.	1. Les fondations.	1. Die Grundmauer.
2. Hearth-level.	2. La sole.	2. Die Herdsohle; die Ofensohle.
3. Base-plates.	3. La base des colonnes.	3. Der Säulenfuss.
4. Columns.	4. Les colonnes.	4. Die Säulen.
5. Mantel beams.	5. Le manteau en tôle.	5. Das Manteleisen; der Tragkranz.
6. Mantel-plate.	6. Le support du manteau.	6. Die Mantelplatte; der Eisenkranz.
7. Shaft.	7. La cuve.	7. Der Schacht.
8. Boshes.	8. Les étalages.	8. Die Rast.
9. Throat.	9. Le gueulard.	9. Die Gicht.
10. Bustle-pipe bracket.	10. La console de la conduite de vent.	10. Der Windleitungsträger.
11. Bustle-pipe hanger.	11. Le support de la conduite de vent.	11. Das Hängeisen der Windleitung.
12. Bustle-pipe saddle.	12. L'étrier du conduit d'air.	12. Der Traglappen der Windleitung.
13. Bustle-pipe	13. La conduite de vent.	13. Die Windleitung.
14. Bosh-plates.	14. Les plaques supportant les étalages.	14. Die Rastkühlplatten.
15. Hearth.	15. L'ouvrage.	15. Das Gestell.
16. Crucible.	16. Le creuset.	16. Das Untergestell; der Herd.
17. Tymp.	17. La tympe.	17. Der Tümpel.
18. Front or tymp-arch.	18. La voute de la tympe.	18. Das Tümpelgewölbe.
19. Dam.	19. La dame.	19. Der Dammstein; der Wallstein.
20. False or monkey-dam.	20. La fausse dame.	20. Der falsche Dammstein.
21. Tuyere.	21. La tuyère.	21. Die Windform.
22. Nozzle.	22. La buse.	22. Die Düse.
23. Slag or Cinder-notch.	23. Le trou des laitiers.	23. Die Schlackenform.
24. Slag or Cinder-notch arch.	24. L'embrasure du trou des laitiers.	24. Das Schlackenloch.
25. Slag or Cinder-notch runner.	25. Le conduit des laitiers.	25. Die Schlackenrinne.
26. Hearth-jacket.	26. Le manteau pour le creuset.	26. Der Herdmantel; die Gestellverankerung.
27. Tap-hole.	27. Le trou de coulée.	27. Das Stichloch; der Stich.
28. Iron-runner.	28. Le conduit de coulée.	28. Die Eisenrinne; das Gerinne.
29. Crinoline-bands.	29. Les cercles en fer.	29. Die Eisenbänder; die Rastbänder; die Anker.

ENGLISH.	FRENCH.	GERMAN.
30. Waste-water pipe.	30. Le tuyeau de decharge des tuyeres.	30. Das Wasserabflussrohr.
31. Feed-water pipe.	31. Le tuyau d'alimentation.	31. Das Speisewasserrohr; das Wasserzuflussrohr.
32. Belly-pipe.	32. La conduite d'eau des tuyères.	32. Das Düsenrohr.
33. Tuyere-stock.	33. Le porte-vent.	33. Der Windstock.
34. Ventilating-ports.	34. Les trous d'aérage.	34. Die Dunstschlitze; die Abzugsöffnungen.
35. Bosh-ports.	35. Les trous d' aérage des étalages.	35. Die Dunstschlitze der Rastform.
36. Bosh-tuyere.	36. Les tuyères des étal ages.	36. Die Rastform.
37. Bleeder.	37. Le tuyau de déchapment des gaz.	37. Das Gasabflussrohr; das Gasauslassrohr.
38. Bleeder-valve.	38. Le clapet d'echapment des gaz.	38. Das Register des Gasabflussrohres; der Verschluss des Gasauslassrohrs.
39. Down-comer.	39. Le conduit de gaz.	39. Das Gasableitungsrohr.
40. Dust-catcher.	40. La poche aux poussières.	40. Der Staubsack.
41. Dust-catcher gooseneck.	41. Le col de cygne pour la poche aux poussières.	41. Der Hals des Staubsacks.
42. Dust-catcher bell.	42. Le cone de la poche aux poussières.	42. Der Glockenverschluss des Staubsacks.
43. Dust-catcher columns.	43. Les supports de la poche aux poussières.	43. Die Säulen des Staubsacks.
44. Feed-water - pipe bracket.	44. La console du tuyau d'alimentation.	44. Der Träger des Speisewasserrohres.
45. Waste-water-pipe hanger.	45. Le support du collecteur des eaux chaudes.	45. Das Hängeisen des Abflussrohrs.
46. Furnace-shell.	46. Le revêtement en tôle.	46. Der Blechmantel.
47. Bell.	47. Le cone.	47. Der Kegel; die Glocke.
48. Hopper.	48. La trémie.	48. Der Trichteraufsatz.
49. Hopper extension-ring.	49. L'anneau inférieur de de la trèmie.	49. Die Trichterverlängerung.
50. Hopper lip-ring.	50. Le rebord supèrieur de la trémie.	50. Die Unterstützung des Trichters.
51. Top platform.	51. La plateforme du gueulard.	51. Der Gichtbelag.
52. Platform-brackets.	52. Les consoles du gueulard.	52. Die Träger des Gichtbelags.
53. Hopper-brackets.	53. Les consoles de la trémie.	53. Die Träger des Trichters.

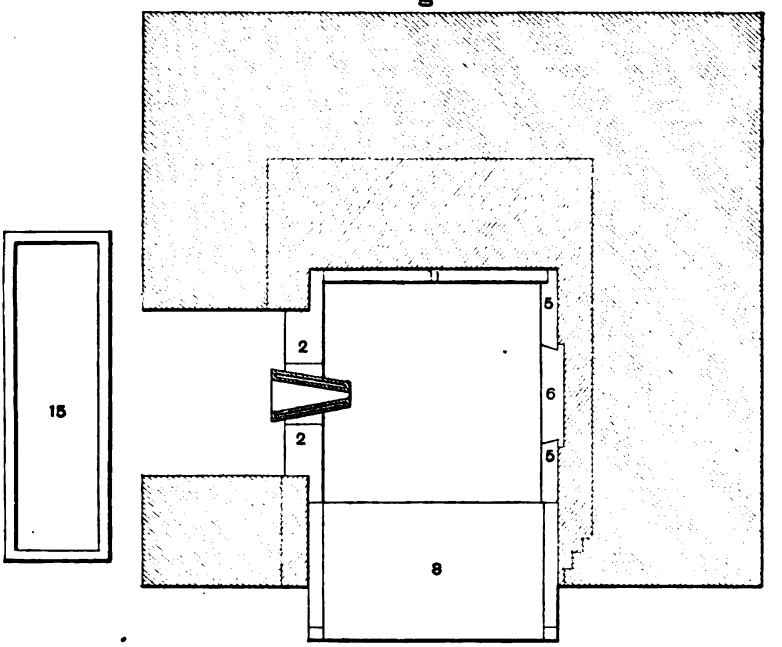
ENGLISH.	FRENCH.	GERMAN.
54. Bell-arch.	54. Le support du cone.	54. Die Unterstützung der Glocke.
55. Bell-beams.	55. Les leviers de manœuvre du cone.	55. Die Hebel der Glocke.
56. Bell lifting-rod.	56. La tige de suspension du cone.	56. Die Hubstange der Glocke.
57. Bell safety-rods.	57. La tige de sûreté du cone.	57. Die Sicherheitsstangen der Glocke.
58. Bell counter-weights.	58. Les contrepoids du cone.	58. Die Gegengewichte der Glocke.
59. Bell-winch.	59. Le treuil de manœuvre du cone.	59. Die Winde der Glocke.
60. Bell air-cylinder.	60. Le cylindre pneumatique du cone.	60. Die Luftpumpe der Glocke.
61. Waste-gas flume.	61. La cheminée d'echappement des gaz.	61. Der Gasableiter.
62. Furnace-canopy.	62. Le toit du fourneau.	62. Das Dach des Ofens; das Gichtdach.
63. Bridge-beams.	63. Les poutres du pont.	63. Die Träger der Gichtbrücke.
64. Bridge-roof.	64. Le toit du pont.	64. Das Dach der Gichtbrücke.
65. Platform-guards.	65. Le parapet.	65. Die Schutzbleche.
66. Throat- or stock-line.	66. Le niveau du gueulard.	66. Die Gichtebene; die Gichtöffnung
67. Bosh.	67. Le ventre.	67. Der Kohlensack.
68. Height.	68. Le hauteur du four.	68. Die Höhe des Ofens.
69. In-walls.	69. Les paroirs refractaires.	69. Der Kernschacht.
70. In-walls backing.	70. La chemise intermédiaire.	70. Die Hintermauerung des Kernschachtes; der Zwischenschacht.
71. In-walls filling.	71. La remplissage des murs.	71. Die Fülluqg der Schachtmauerung; das Rauhgemauer; der Rauhschacht.
72. Bosh-walls.	72. Les paroirs des étalages.	72. Die Rastmauerung.
73. Crucible-walls.	73. Les paroirs du creuset.	73. Die Mauern des Untergestells; die Herdmauern.
74. Hearth-blocks.	74. Les briques de la sole.	74. Die Bodensteine.
75. Expansion-space.	75. L'intervalle pour l'expansion.	75. Der Schachtausdehnungsraum.
76. Explosion-door.	76. La soupape de sureté des gaz.	76. Die Sicherheitsklappe; die Explosionsklappe.
77. Bustle-pipe man-holes.	77. Trou d'homme du conduit d'air.	77. Das Mannloch der Windleitung.

Fig. 1.



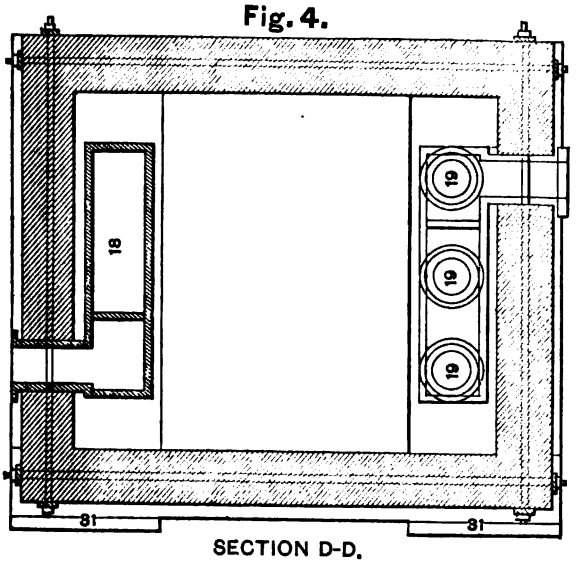
The Bloomary.

Fig. 2.



SECTION A-A.

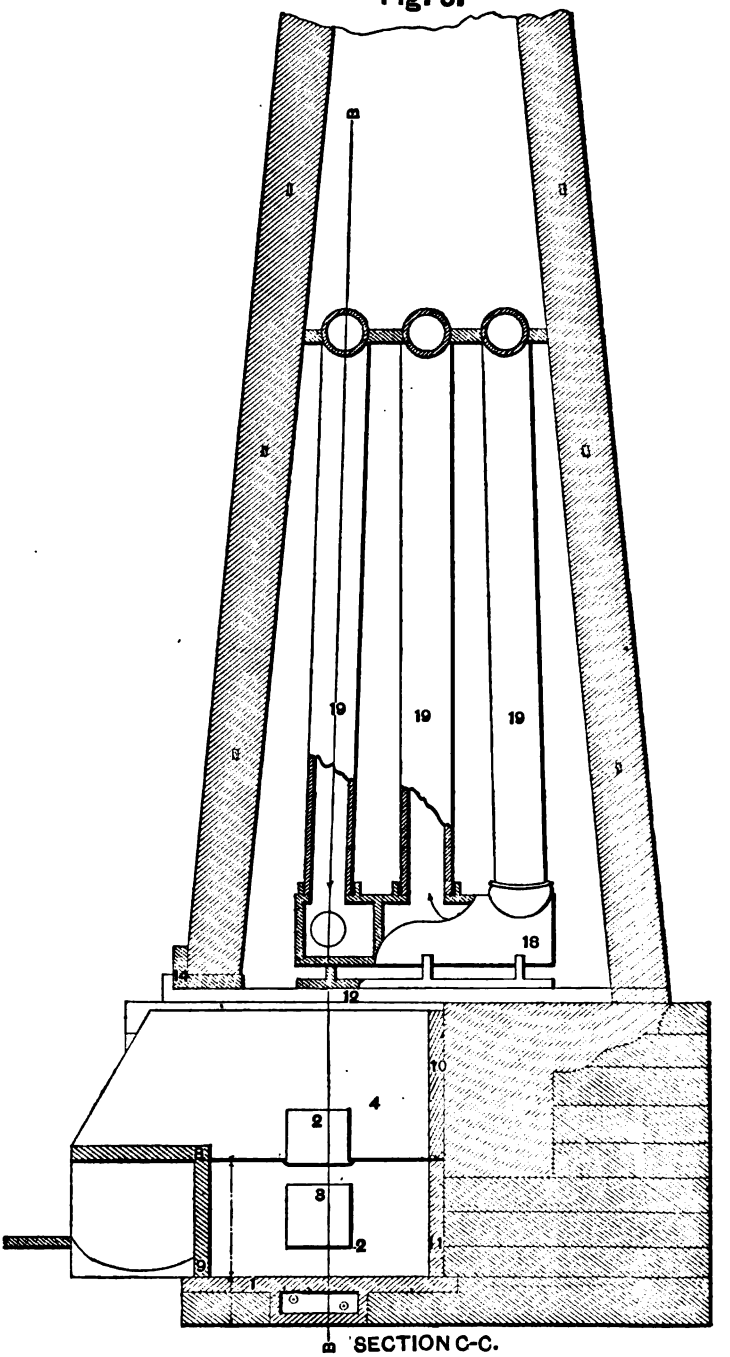
Fig. 4.



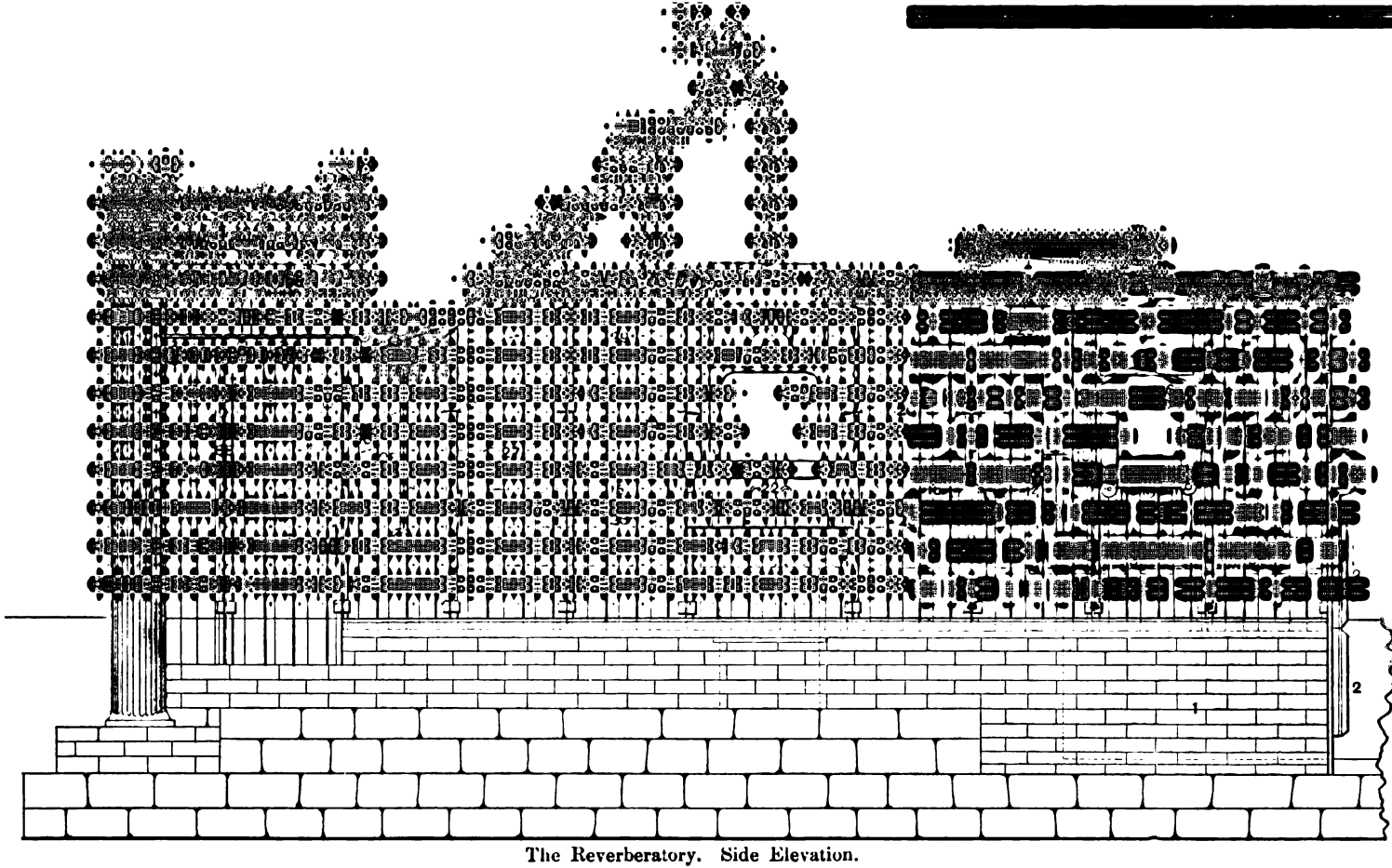
SECTION D-D.

The Bloomary (*Continued.*)

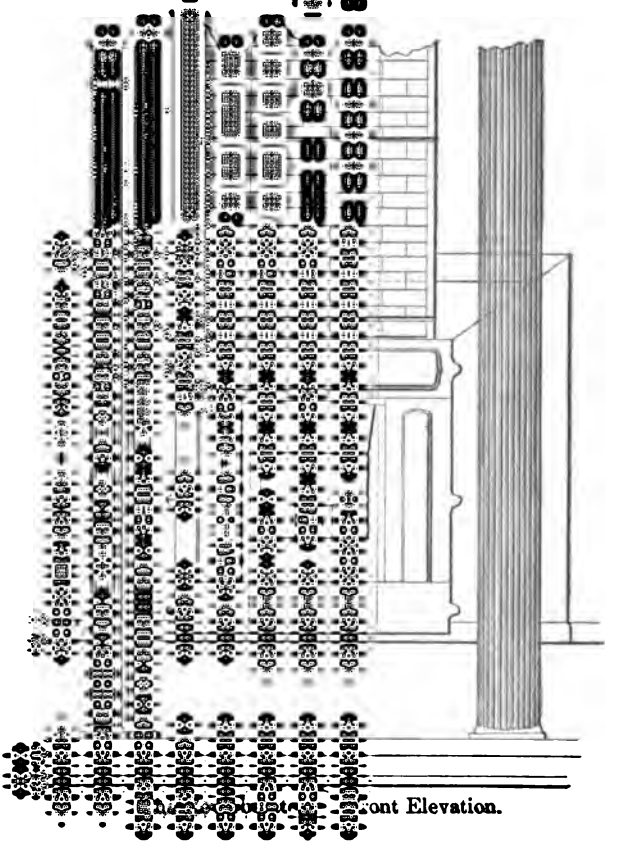
Fig. 3.



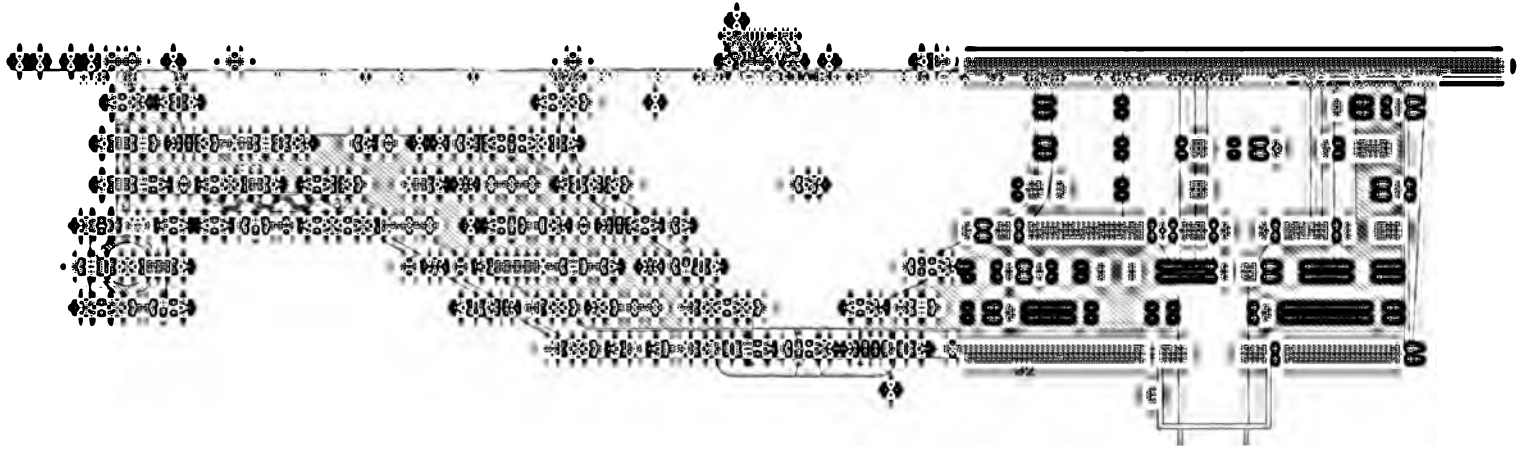
The Bloomary (*Continued.*)



The Reverberatory. Side Elevation.

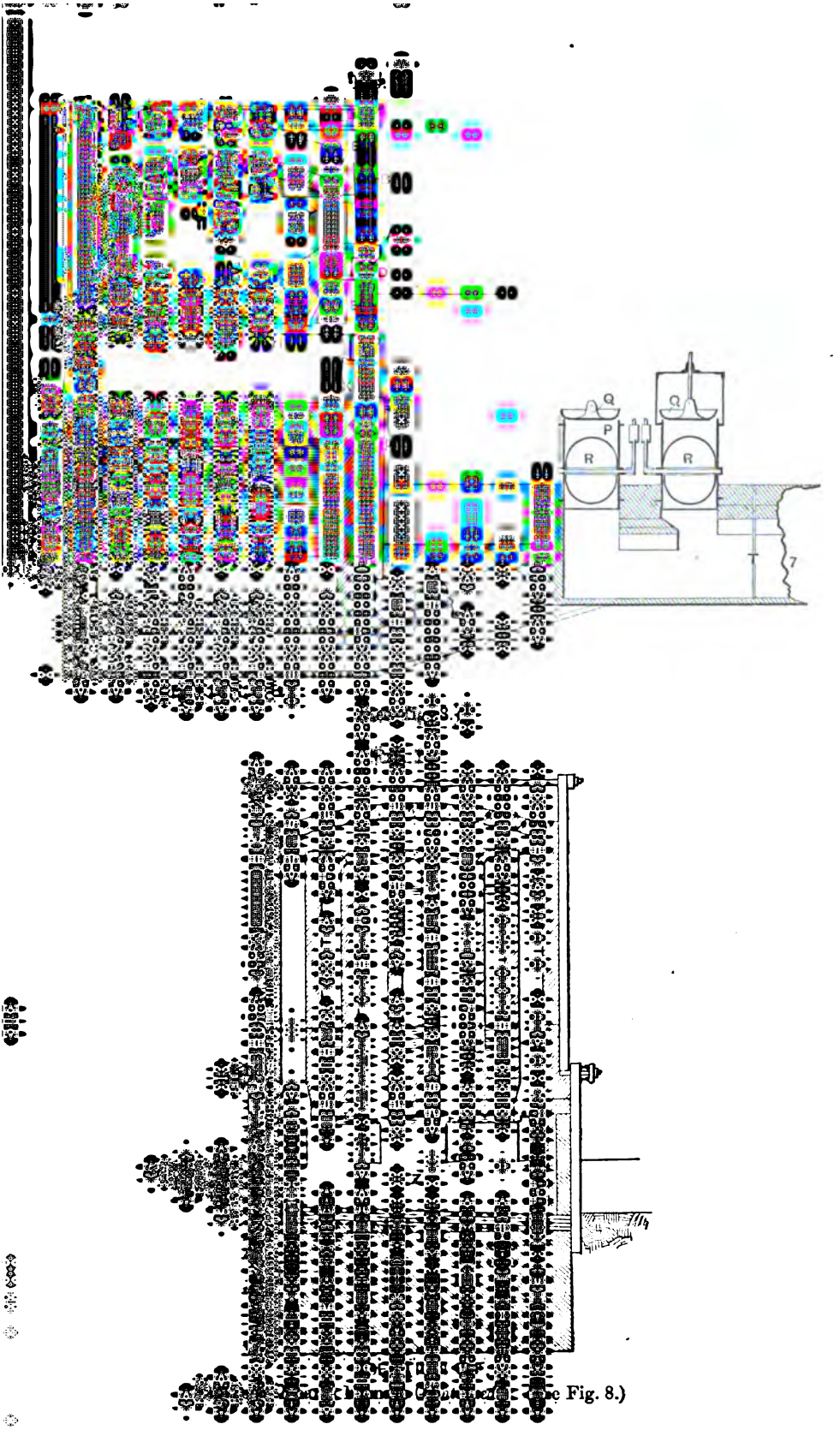


Three Quarters Front Elevation.



The Reve·beratory. Half Plan.

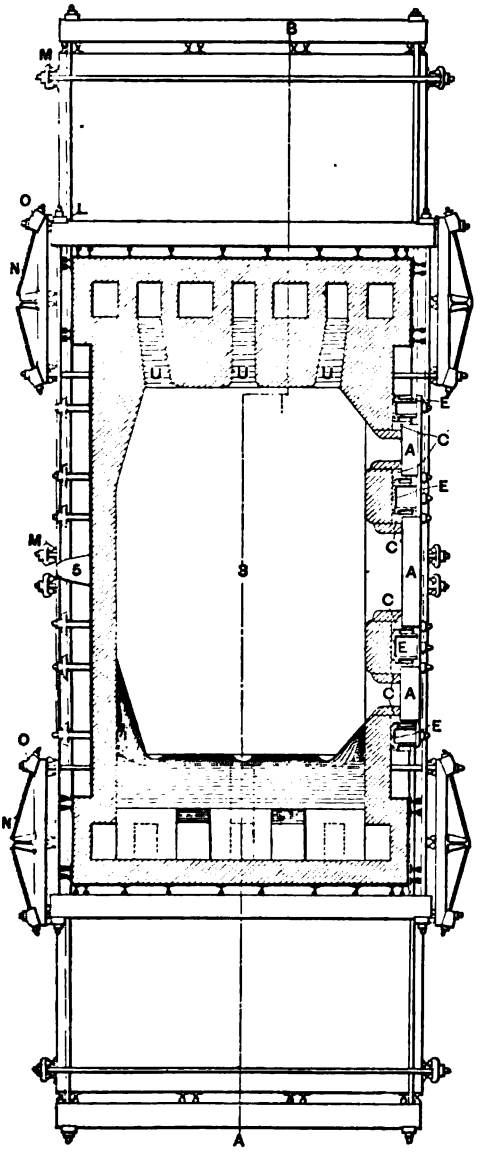
The Open-Hearth Furnace. (See Fig. 12.)



e Fig. 8.)

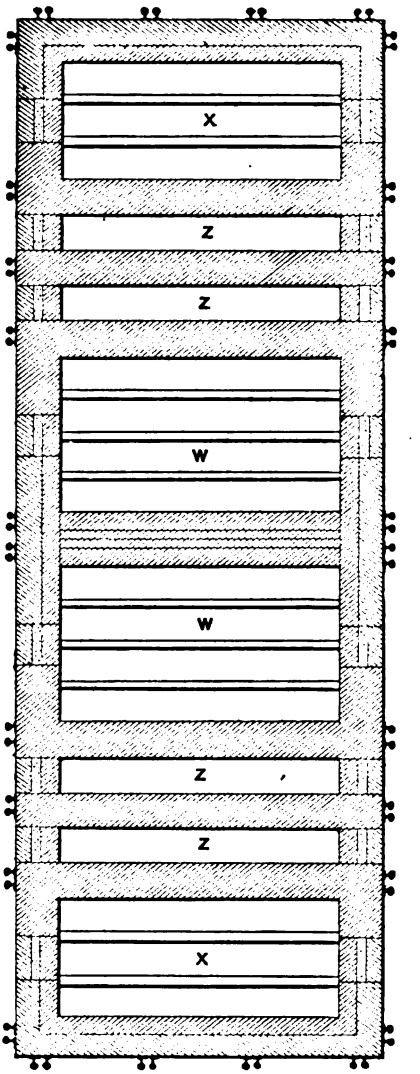
(*Continued*). (See Fig. 8.)

FIG. 12.



SECTION J-K.

FIG. 13.



SECTION L-M.

The Open-Hearth Furnace (*Continued*). (See Fig. 8.)

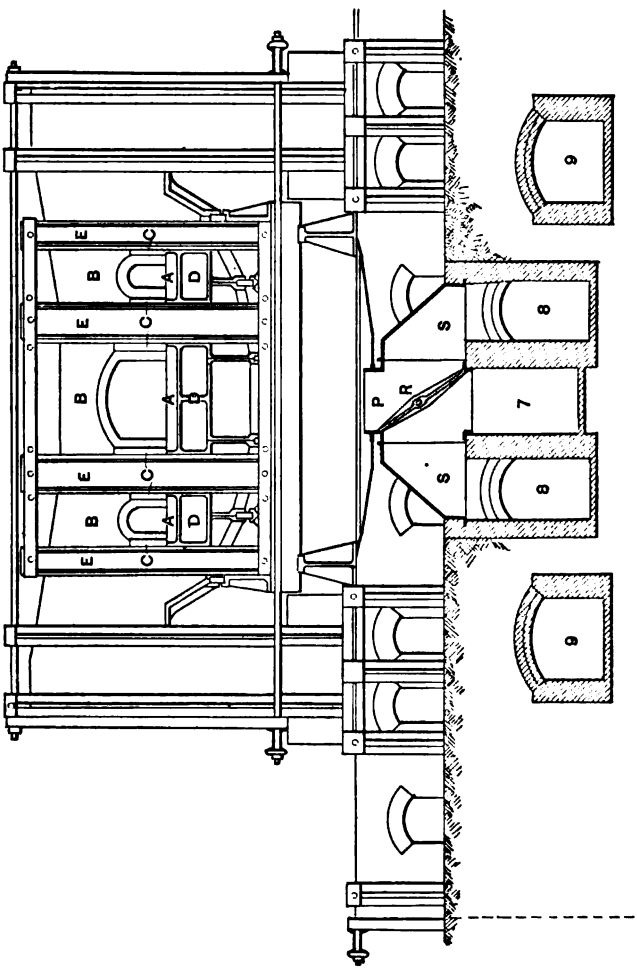
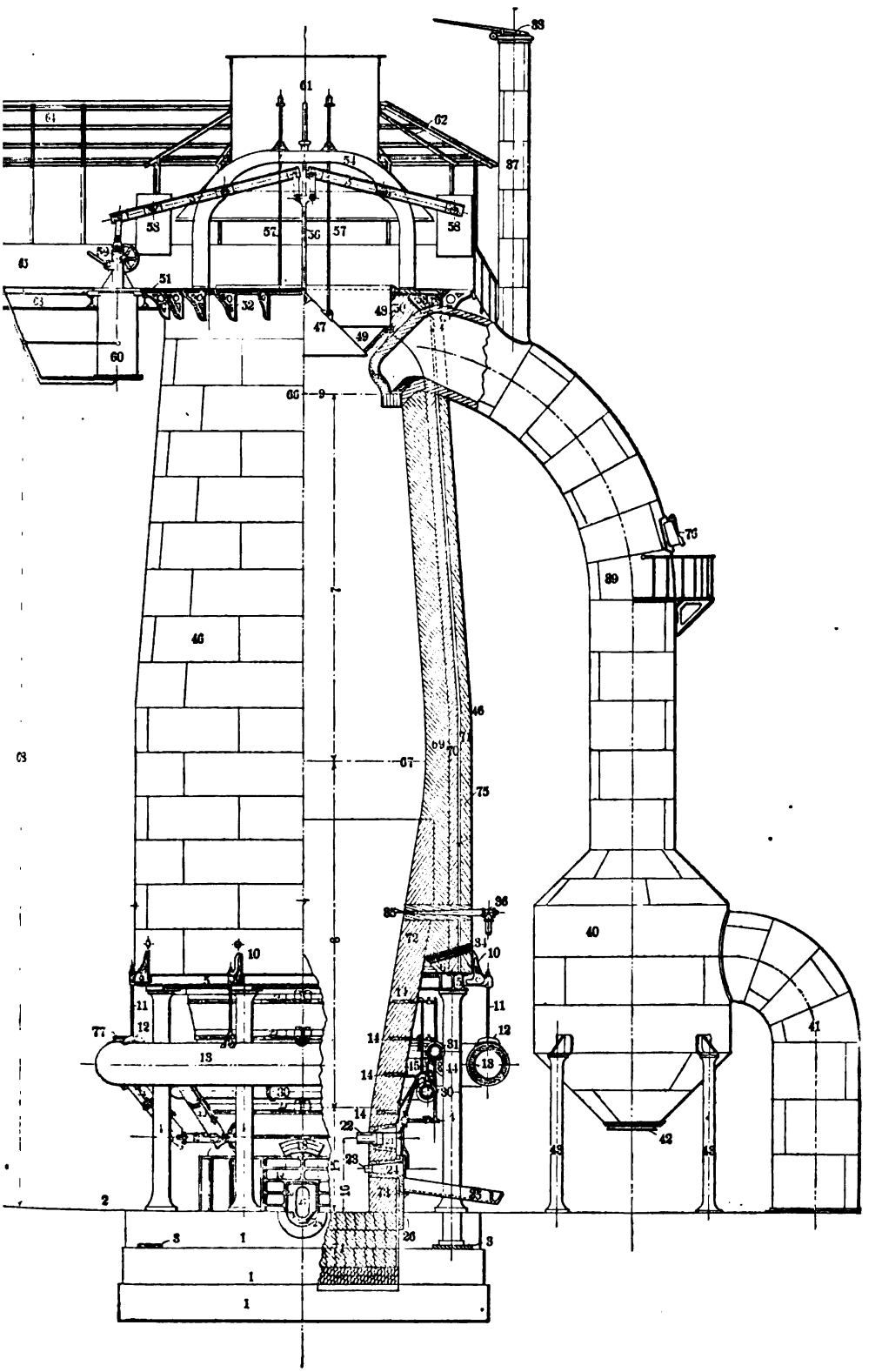


FIG. 14.

SECTION N-O.

The Open-Hearth Furnace (*Continued*). (See Fig. 11.)

FIG. 15.



The Blast-Furnace.

HUSGAFVEL'S IMPROVED HIGH BLOOMARY FOR PRODUCING IRON AND STEEL DIRECT FROM ORE.

BY F. LYNWOOD GARRISON, PHILADELPHIA, PA.

(Boston Meeting, February, 1888.)

EXCEPT in the old Catalan forge, or its modifications, attempts to make iron and steel directly from ore in a practical and economical manner have failed so frequently and completely that such schemes are looked upon with justifiable distrust. Without exception, all have failed from one cause or another, although in some instances every indication of a practical success was manifest in the beginning. It is, therefore, necessary to carefully examine and consider every detail affecting the process, or more properly, furnace, under consideration. This I have done to the best of my ability during a long visit to Russia last summer, and after satisfactorily observing the practical success of the furnace in the iron works of Count Stroganoff at Dobriansky in the Government of Perm, I decided to investigate as completely as possible the history of similar furnaces in Finland and Russia. These data, sifted and verified with the greatest possible care, form the substance of the present paper.*

An illustration on page 321 of Percy's *Iron and Steel* shows the old type of *Osmundssmide* or *harkugnar*, which has been in use in Northern Europe since pagan times. Fig. 1 shows a furnace of this kind. It should, perhaps, more properly be called a high-bloomary furnace or forge, the Swedish word *hark* meaning *bloom* or *loupe*. It was with this type of furnace that Mr. Husgafvel began his experiments at Porsaskoski in 1875. Attempts to obtain better results with larger furnaces of this type were failures, as every bloom or loupe obtained necessitated blowing out the furnace with a consequent large waste of fuel. A larger furnace of course increased this waste with no corresponding advantage. A movable hearth of three-eighth inch iron, lined with a mixture of clay, quartz and lime, was next tried, but in consequence of the frequent repairs of this lining,

* See Russian *Mining Journal*, vol. ii., 1887, pp. 398 to 435. Husgafvel's paper in *Jernkontorets Annaler*, 1887. Also, *Journal of United States Association of Charcoal Iron Workers*, vol. vii., No. 5, pp. 280–306.

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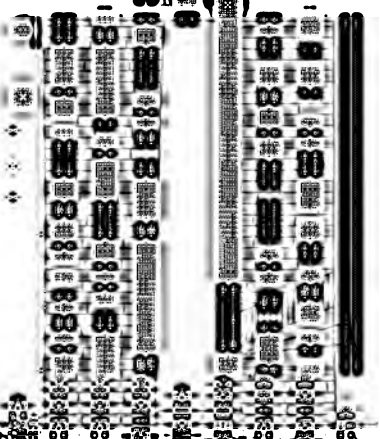

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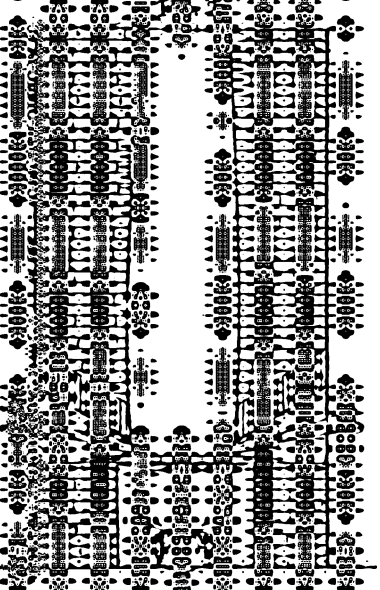

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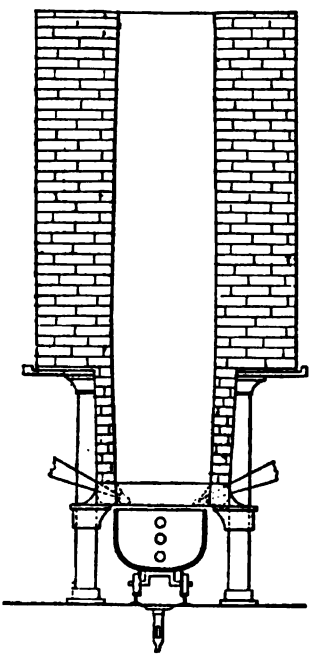
l, but the product

what similar to this, is

was not homogeneous, as the reduced iron stuck to the brick walls of the furnace, obstructing and rendering the operation irregular; and at times the temperature in the lower part of the furnace became so high that pig-iron was formed. To avoid these difficulties and to decrease the temperature of the blast the walls were made thinner (4½ inches) and the furnace was provided with a hollow cast-iron ring immediately above the hearth, through which the tuyeres, cooled with water and movable in all directions, were inserted. The furnace thus altered, as shown in Fig. 3, was erected in Pankakoski,

FIG. 3.



Improved Furnace, built at Pankakoski, 1880.

Finland, in the year 1880. In the beginning the hollow cast-iron ring was cooled with water, but it was soon found that such cooling was unnecessary and did not produce the desired result; besides, the circulation of the outside air was found to be sufficient.

The results obtained with this furnace were superior to any previous ones; the iron seldom stuck to the walls and the blooms or loupes became more homogeneous. Nevertheless, irregularities in its operation could not be avoided. To make it possible to regulate the changes of temperature caused by heterogeneous materials, etc., and at the same time to cool down the lower part of the furnace, its

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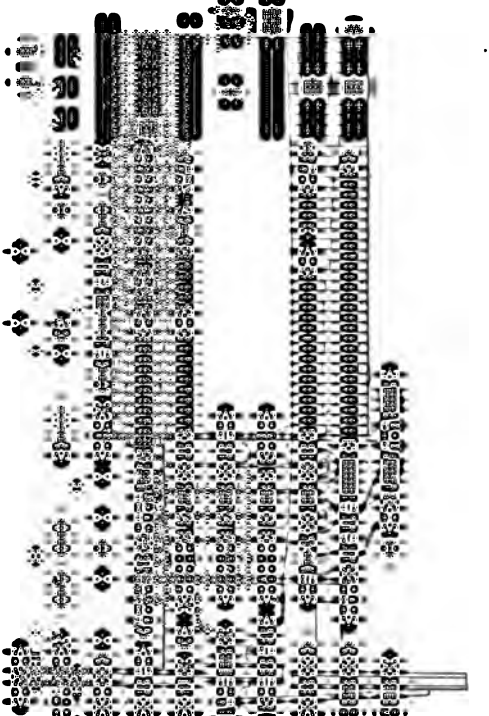

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walls, and although the temperature in the lower part of the furnace was comparatively low (scarcely dark-red) a good welding-heat was nevertheless predominant in the hearth. It worked, however, with only fair regularity, and the resulting loupes were not homogeneous and contained much slag. It was discovered that the tuyeres were too high above the bottom of the hearth and the blast did not have sufficient effect in all directions. The furnace, therefore, had but a short existence, and in 1883 it was blown out in consequence of a general want of water at the works. To make the blast hotter, and, when desired, to alter its temperature, Mr. Husgafvel proposed to make the entire body of the furnace with double walls of wrought-iron plates, so arranged that the cold-blast could be admitted at different levels, his object being to enable alterations of temperature to be made in the furnace itself, thus regulating the operation of the smelting or reducing process. In the year 1884 these principles were applied, and a furnace with double walls of soft iron-plate was erected in the Wärtsilä iron-works, located in the eastern part of Finland near Lake Ladoga.* The results obtained with this furnace (Figs. 5 and 6) showed that the productive capacity increased in the same ratio as the size of the furnace, at the same time effecting a great saving of fuel, although not quite in the same proportion.

Comparing the results of this furnace with the oldest (Fig. 1), we find that the furnace whose volume is eight times that of the latter, produced eight times as much more, and that the expenditure of charcoal, which in the old furnace was 280 units of weight per 100 units weight of blooms, was in the Wärtsilä furnace only 105 units; furthermore, the expenditure of ore decreased from 536 to 313 units weight of ore to 100 units weight of blooms (loupes).

Mr. Husgafvel attributes these favorable results to the peculiar circulation of the blast through the hollow walls of the furnace and to the insertion of the tuyeres into the hearth itself, the former preventing the too early sintering of the ore, and the sticking of the charge to the walls. By a peculiar arrangement of charcoal in the hearth the product was made more uniform and at the same time an undue amount of oxidation prevented. The charcoal used in all the preceding furnaces was made of ordinary fir and birch. In the autumn of 1885 the charcoal charges were replaced with the same weight of English coke, and it is reported that no especial change in the ope-

* See description in Russian *Mining Journal*, No. 11, 1886. Also, paper on "New Material for the Open-Hearth Process," same journal, No. 9, 1885.

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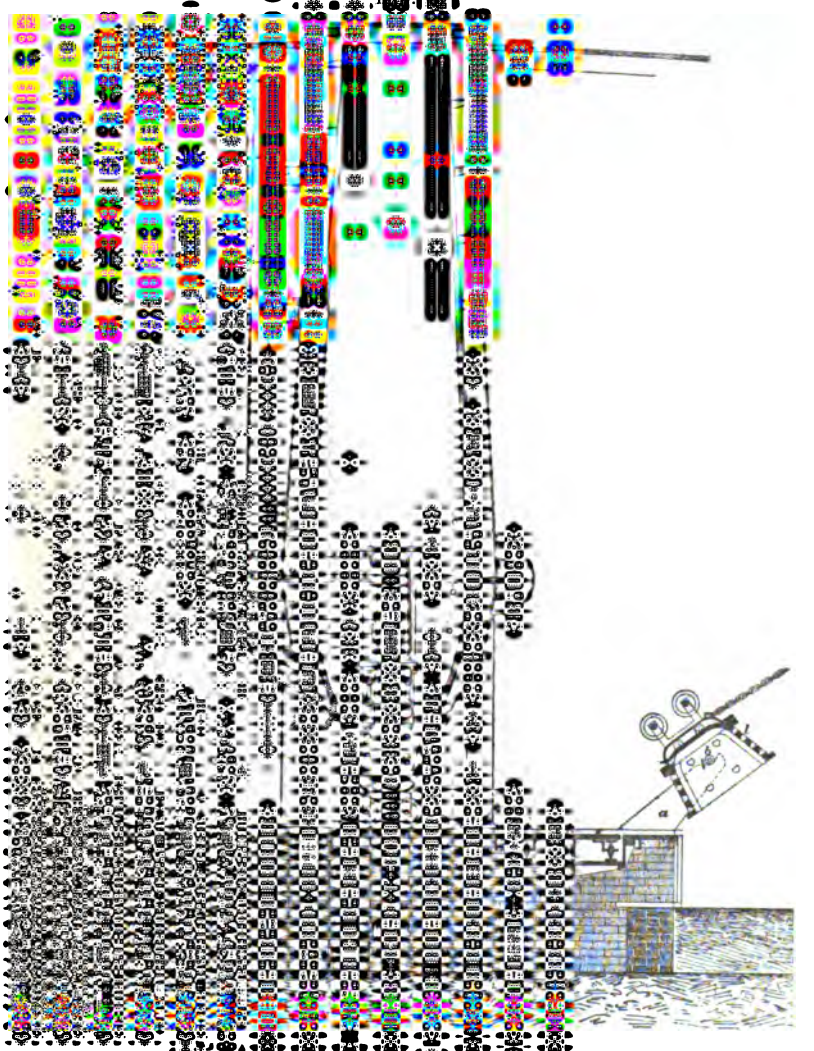
should be observed
in no case intended



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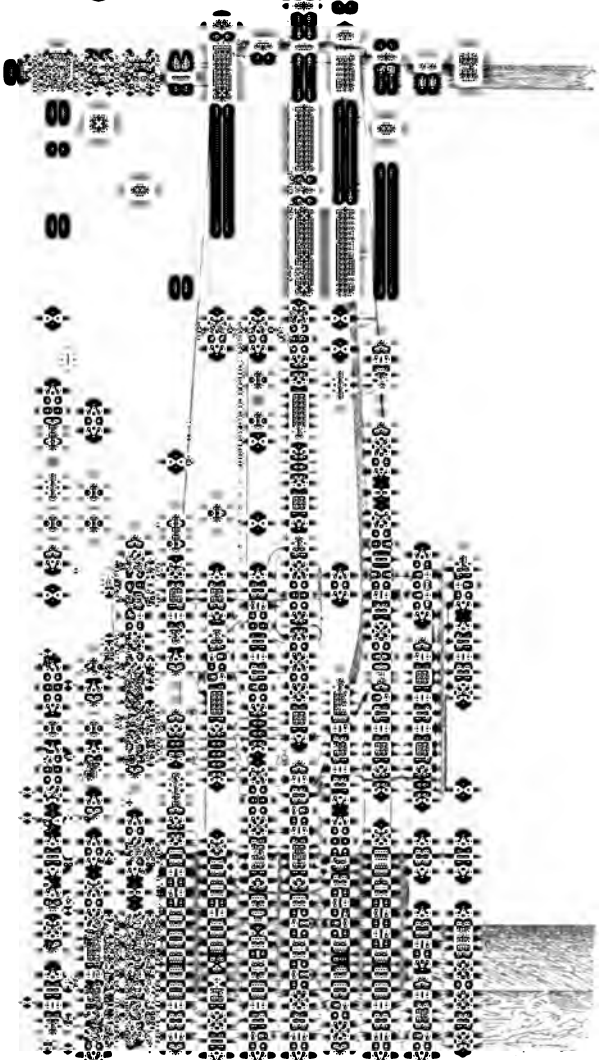
phosphorus and the loupe non-homogeneous and cold-short. Further particulars upon these points will be given further on.

The improvements made in the Wärtsilä furnace can be readily seen by comparing Fig. 5 with the preceding furnaces. The walls of the Wärtsilä furnace are of wrought-iron plate, the whole being enveloped by a second wall or shell, extending from the hearth to the throat, thus forming a series of zones, divided by spiral partitions, making a continuous spiral flue, coiled, as it were, around the furnace. The blast, passing through this, is heated under ordinary circumstances from 150°–250° C. (302°–482° F.), though sometimes running up much higher. By means of a system of dampers *t* and the pipes r, r', r'', r''', r'''', Fig. 5, the circulation of the blast through these chambers or flues can be regulated with the greatest ease. If the blast is too hot, the upper and middle chambers are shut off; if, on the contrary, it is too cold, the upper chambers, which are frequently not used at all, are opened. As can be easily understood, this contrivance, together with the movable hearth, gives a most effectual control over the operation of the furnace. To check loss of heat by radiation, the outer shell is coated on the outside with a four-inch layer of fire-clay. The other most important improvement made in the Wärtsilä furnace was the insertion of the four tuyeres directly into the movable hearth. There are four tuyere-holes *c*, in each of the two opposite sides of the hearth, the lower being 3 inches and the upper 6 inches from the bottom. When a fresh hearth has been inserted and a new loupe is beginning to form, the tuyeres are inserted in the lower holes; as the loupe increases in size their inclination is lessened, the object being that the blast should just impinge upon the surface of the loupe. When the loupe has grown up nearly to the level of the lower tuyere-holes, the tuyeres are removed, placed in the upper holes, and the lower ones stopped with clay. The same operation is then repeated until the loupe reaches the level of the upper holes, when the tuyeres are taken out, and the hearth is removed and replaced with a fresh one. Thus the operation is practically continuous, the blast being stopped for about five minutes while the hearth is being replaced. The hearth stands upon a vertically moving platform, which can be raised and lowered a few inches, thus permitting a tight joint being made between the furnace proper and the hearth, the crack, of about an inch, between the two, being first carefully luted with clay before the hearth and platform are raised. The slag is tapped off into a car running under the hearth, by a series of four holes (*l*) one above the other. The hearth con-

ther, is revolved on
in Fig. 5. If the
the car beneath, a
e knob, x (Fig. 7),



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the tuyeres is compli-
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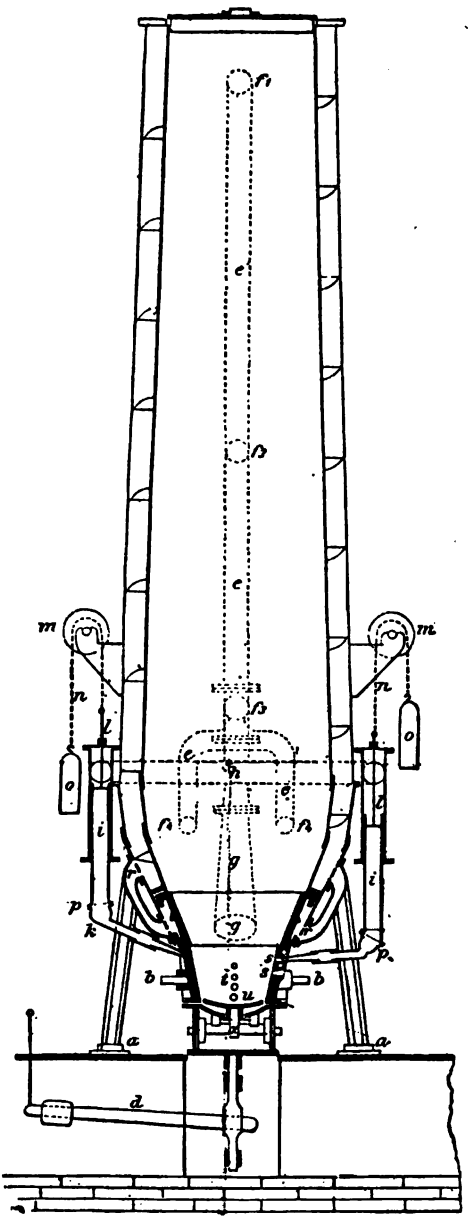
FIG. 7.



Dobriansky Furnace, 1886. Vertical Section.

furnace. Through the tube *i* it goes down to the tuyere-stocks *k*, two on each side. These tubes and stocks can be moved vertically by means of the rods *l* and chains *n*, provided with counter-balance *o*, passing over the wheel *m*. By means of ball and socket-joints at *p* the tuyeres are movable in any direction; each tuyere is provided with a separate stop-cock, that the blast may be shut off from it if desired.*

The Production and Economy of the Wärtsilä Furnace.—The following details are mostly derived from official reports and certificates of the Wärtsilä iron-works.

The furnace produced in twenty-four hours from puddle-slags, lake- and bog-ores the following amounts of *malleable* iron.†

Charges consisting of:	Weight of Iron Produced.‡		
	Min.	Max.	Mean.
Lake-ores (net tons)	2.10	2.85	3.00
60 per cent. Lake-ores and 40 per cent. puddle slag	2.24	3.37	2.64
Puddle slag	.....	.....	1.52

The following table shows the relative amounts of material required to produce one net ton of malleable iron (loupe).

‡Amounts of material required to produce one ton (net).	A.			B.		
	Min.	Max.	Mean.	Min.	Max.	Mean.
⁂Charcoal (bushels)	138	181	159	138	177	157
Lake-ores (tons)	2.55	3.35	2.87	1.40	1.89	1.64
Puddle-slags	.....	.....	.....	0.95	1.10	1.02
Burnt lime for hearths	.....	.....	0.0109	.....	.....	.0109
Limestone	.....	.....	0.078	.....	.....	0.121
Cast-iron for repairs of hearths	.....	.....	0.006	.....	.....	.010

The charcoal used was made from a mixture of fir and pine; quality, medium. The lake-ores contain about 12 per cent. hydroscopic

* This description applies to the tuyere construction used on the *Dobriansky* furnace; it is inserted here as it is more easily understood than that of the Wärtsilä furnace.

† Official certificate of managers, May 6th, 1885.

‡ These weights were originally in Russian poods, 1 pood = 36.10 pounds.

⁂ This amount of charcoal includes that used for preparatory heating and for filling the hearths.

moisture, 15 per cent. SiO_2, and from 0.31 to 0.76 per cent. phosphorus. These same ores yielded in the Wärtsilä charcoal blast-furnace, making pig-iron, from 34.1 to 37.8 per cent. pig.* An average sample of puddle slag, finely ground but not roasted, contained:

SiO_2	= 16.00
Al_2O_3	= 6.00
MuO	= 5.70
Phosphorus	= 4.00
CaO	= 4.00
Iron	= 49.00

The Husgafvel furnace required repairs after producing 19,000 poods (342 net tons) of loupe, the cost of which repair amounted to only about 40 Finnish marks ($8.00).† The smelting process is managed by one smelter with assistant, and one charger for each shift of twelve hours. Separate workmen are sometimes employed to assist in charging, removing slag, etc. The cost of the loupe as it comes from the movable hearths is about equal to charcoal pig-iron made from the same ores. The following table of comparative cost of pig-iron and blooms (loupe) at Wärtsilä is translated from the official report of the engineer sent by the Russian Government to investigate the Finnish iron industries.‡ He says, "It cannot be without interest to compare the economical results of the working of the Wärtsilä furnace with the working of the blast-furnace in the same place, both using ores of the same chemical composition."

	Gray Pig Iron.			Husgafvel Blooms (Loupe).		
	Weight or Measure.	Penni.ǂ	Kopeck.ǂ	Weight or Measure.	Penni.	Kopeck.
	Pood.ǂ					
Ores	2.77	55.4	13.85	2.92	58.6	14.65
Puddle Slags	.....	.....	.....	0.609	2.13	0.53
	Pood.ǂ					
Fluxes	0.35	2.1	0.52	0.141	0.84	0.21
	Korob.ǂ					
Charcoal	0.066	40.00	10.00	0.076	45.90	11.47
Wages	.....	9.0	2.25	.....	14.0	3.50
Repairs, etc.	.....	24.0	6.00	.....	9.00	2.25
		130.5	32.62		130.47	32.61

* This furnace is 42 feet high and 2646 cubic feet capacity.

† Official Reports.

‡ Wasileffski in Russian *Mining Journal*, February 1887, page 307.

ǂ Russian Pood = 36.10 pounds, korob = 43 bushels, kopeck = about ½ cent, Finnish penni = ¼ cent.

"For each korob of charcoal 15 poods 6 pounds (Russian) of pig-iron were produced, and with the same amount of charcoal 13 poods 5 pounds of Husgafvel blooms" (loupes).

From a number of different analyses the carbon and phosphorus in the Husgafvel loupe averaged as follows: phosphorus, 0.29 to 0.85 per cent.; carbon, 0.12 to 2.00 per cent. This material reheated and made into billets and blooms contained: phosphorus, 0.18 to 0.77 per cent.; carbon, 0.10 to 1.20 per cent.* From two-thirds to eight-tenths of the phosphorus in the charge can be eliminated with the slag. The furnace can also be so managed that a more or less carburetted product can be obtained at will, and generally such a product is nearly homogeneous. In the following table the prices of materials and labor in Wärtsilä, Finland, are given for the year 1885:

Charcoal per bushel,†	17 penni	=	3⅓ cents.
Lake-ores per ton,†	12 marks	=	$2.40.
Puddle-slags per ton,	1.70 marks	=	34 cents.
Burnt lime per ton,	20 marks	=	$4.00.
Day's work for one man,	1.80 marks	=	36 cents.
Day's work for one man and horse,	2.40 marks	=	48 cents.

From the best information obtainable outside of official reports, I have calculated the cost of the loupe, as produced in the Husgafvel furnace as 1.40 marks (Finnish), or 28 cents per pood, about equal to $16.25 per ton. In the official report of the Russian engineer Wasileffski it is estimated at 1.30 marks, or 26 cents per pood, equal to about $15.60 per ton. As the difference between these two estimates is so small, it is probably safe to conclude both are approximately correct. It should be borne in mind that these blooms or loupes can only be used in the basic Siemens-Martin furnaces, as they contain too much phosphorus to be hammered and rolled. Had they been made from magnetic ores, low in phosphorus, their cost would have been considerably increased, on account of the more refractory nature of such ores.

The first furnace designed for the use of hard refractory ores, and to make loupe capable of being directly hammered and rolled without further refining, was started June, 1886, in Count Stroganoff's iron-works at Dobriansky, in the Ural, and has given great satisfaction ever since. (See Figs. 7 and 8). It was originally intended to build two furnaces, 35 feet in height, 9 feet of which was for gas "take-off." In order to facilitate the building, however, it was deter-

* Official Reports.

† Reduced from Finnish measures.

mined to build only one stack, and to dispense with the "take-off." This resulted in a furnace 26 feet high from the top of the movable hearth to the throat, having a diameter of 5 feet at the belly and 4 feet at the throat. The inner and outer shells were separated by a space of seven inches. It was desirable, in the construction of this furnace, that the deficiencies of the Wärtsilä and other previous furnaces should, as far as possible, be avoided. It had been found that removing the hearths caused a corresponding lowering of the material in the furnace, and that most of the material fell on the side at which the new hearth was introduced, thus causing a slope of the material in the furnace, and a consequent irregularity in its operation. Great difficulty was sometimes experienced in separating the loupe from the hearths, the joints having frequently to be loosened to effect it. The tuyere-walls of the hearth were also a source of annoyance by burning out too rapidly. The greatest of all these difficulties, however, was the sticking of the unreduced iron to the thin brick lining in the lower portion of the furnace. This, happening so near the junction of the stack and the movable hearth caused a deal of trouble. The first difficulty was, in great measure, overcome by introducing the hearths alternately from opposite sides; by this means the unavoidable sloping of the material in the furnace was checked by being made equal on both sides. The tendency of the loupe to stick to the hearth was obviated by providing it with the loose or false bottom before referred to (page 341). In order to check the rapid burning out of the tuyere-holes in the hearths, their sides were cast containing coils of wrought-iron pipe surrounding these holes, and by this means were easily cooled with water. The tendency of the unreduced iron to stick to the lower portion of the furnace was ingeniously overcome by inserting into the lowest part of the stack a hollow cast-iron ring, with its upper edge concentric with, but somewhat less in diameter than, the inner shell, and extending downwards, gradually acquiring a quadratic shape, with rounded edges. The outer shell of the furnace is supported upon flanges, with which this ring is provided. The blast passing through the hollow walls of the furnace is conducted to and passed through this hollow ring by means of two broad flat tubes, situated on opposite sides of the furnace. By this arrangement it is easy to repair the parts of the furnace exposed to a high temperature; and by thus keeping the walls cool at this place, the unreduced iron seldom, if ever, sticks to them. At the same time the temperature of the blast is increased. The inner wall of the furnace is lined with bricks made of

"chamotte,"* 1½ to 2 inches thick, for a height of 5 feet above the upper edge of the cast-iron ring. The upper part of the stack, as well as the cast-iron ring and the hearth, are without any lining. The stack is supported by four pillars made of steel rails fastened together and inserted into cast-iron sockets, the upper ones being fastened to the outer shell of the furnace, and the lower to an iron bed-plate. This bed-plate rests on a foundation of ordinary brick, 2½ feet high, in which two channels are made crossing each other at right angles, thus giving access for the slag-car to be run under the hearth, and also room for the apparatus to raise and lower the hearth.

It has been suggested that the chief reason that the product of the Husgafvel furnace is malleable iron or steel is due to the fact that the charges of ore are much higher per unit of charcoal than in the case of a blast-furnace. When I was at Dobriansky the charges were as follows: For every 100 pounds (Russian) of charcoal, 140 pounds of ore and flux were charged. This ore was magnetic, from the well-known mines of Malo-blagodatj, and was charged in the following proportion:

Ore,	96 per cent.
Limestone,	2 "
Sand,	2 "
	100

When the ore was replaced with roll-scale, 185 pounds of scale and flux were charged per 100 pounds of charcoal. The roll-scale and flux were mixed as follows:

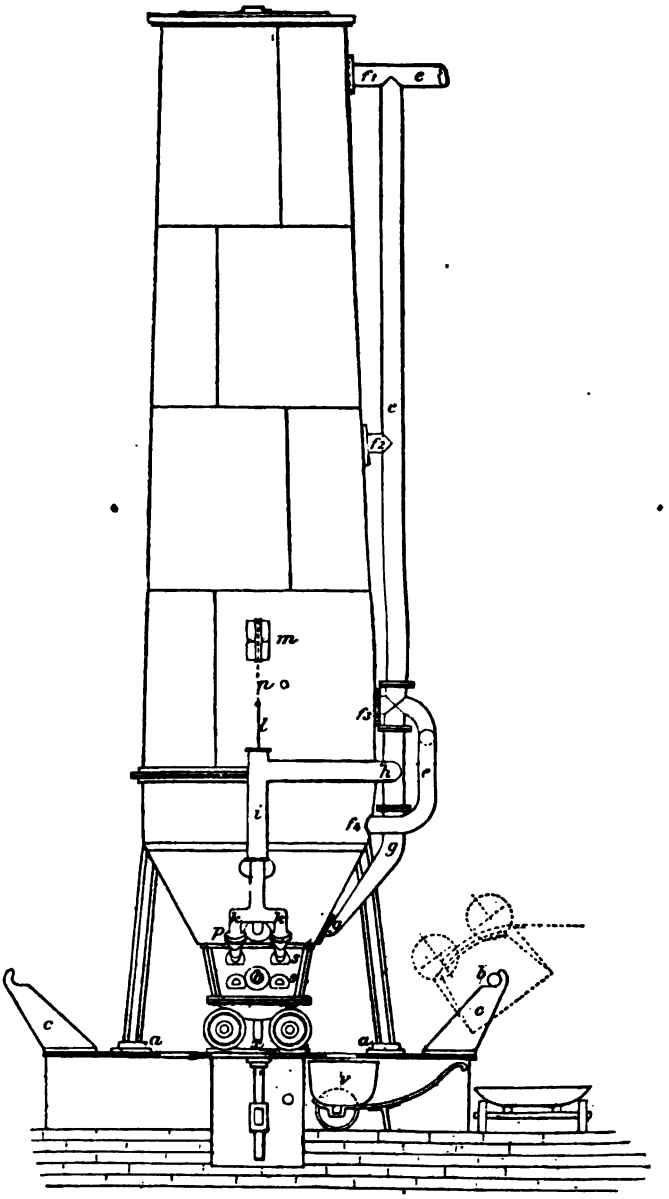
Roll-scale,	95 per cent.
Limestone,	3 "
Dolomite,	1 "
Sand,	1 "

The blast, in passing between the walls of the furnace, checks the reduction of the ore, and, to a great extent, concentrates the reducing action in the lower part of the furnace; consequently, the reduced iron has not sufficient time to take up enough carbon to form pig-iron. As the iron is reduced, it sinks below the thin fluid slag and coagulates together, forming the loupe. During this sinking, a part of the carbon in the iron is oxidized, the extent to which it is thus affected depending, to a great extent, upon the direction given to the blast. A further oxidation may be effected by the ac-

* A mixture of fire-clay and crushed fire-brick.

tion of the blast upon the surface of the bloom. It is usually the case that the loupe is compact and partly solid, like those produced by refining pig-iron in a refining hearth, and not like iron sponge, as might be supposed. The carbonization of the reduced iron can also be regulated by the ore-charges, and the quantity and temperature of the blast. This temperature usually varies from 200° to 235° C. (392° to 455° Fah.); of course, as we have seen, it can be lowered and also raised to a considerable extent if desired. The furnace may be so operated that soft iron or steel can be produced as desired, the only difference in the operation being that in the former case the ore-charges are larger. If hard iron or steel is desired, the temperature in the lower part of the furnace must be increased, and the inclination of the blast into the hearth decreased. The operation of the smelting process can be best judged by the light from the tuyere-holes and the throat-flame; but this should not be relied upon entirely, as perhaps even a better indication is the quality of the slag and loupe, which can easily be ascertained by means of a small iron rod introduced through the tuyere-holes. When the operation is taking place properly the light from the tuyere-holes should be clear and bright, and the throat-flame bright and lively. When soft iron is being properly formed in the hearth, the surface of the loupe should feel hard, even slippery, when touched with a cold iron rod; if the rod becomes hot quickly and particles of iron adhere to its end, it is an indication that the temperature in the hearth is sufficiently high. In a normal operation, the slag should have a light color and flow readily; if, on the other hand, the loupe is hard or uneven, the furnace has been driven too fast, or too much ore has been charged, in which case the slag becomes very quickly fluid and has a yellowish to red color, indicating a great loss of iron. Increasing the degree of carbonization, the slags become thicker and less fluid; at the same time the surface of the loupe becomes soft, according to the degree of the carbonization. The possibility of obtaining a homogeneous product depends upon the regularity of the operation, which, in turn, is due to a uniformity of size and quality of the materials used, together with regular charging and a uniform temperature and quality of the blast. It appears that the greatest difficulties to contend with occur when the charges have been so badly composed that a thick refractory slag is formed. The tuyeres then are apt to grow together, and the throat-flame becomes incipient, bluish, and hot. The remedy of this is, of course, to increase the pressure of the blast, and to introduce it lower down between the

FIG. 8.



Dobriansky Furnace, 1886. Elevation.

walls, so that by a more imperfect reduction the slag becomes richer in iron and more fluid. If, on the contrary, an excess of ore has been charged, the throat-flame seems thin and feeble, the light from the tuyeres less brilliant, and the unreduced ore can be seen through the tuyere-holes; the slag becomes yellowish-red, and, besides, very fluid; the loupe becomes porous and grows rapidly but unevenly. The fuel-charges must then be increased, and the blast introduced higher up and its pressure decreased, until the excess of ore has been overcome. In this, as in all direct processes and indirect refineries, the loss of iron in the slag is considerable. It seems to be dependent upon the quality of the ore and fluxes as much as upon anything else. From the best information obtainable, it seems that the amount of iron in the slags varies from about 18 per cent. when the softest kind of iron is produced, to about 7 per cent. when the product is hard and somewhat steely. When soft iron is produced in the Husgafvel furnace, two-thirds of the phosphorus in the ore may be eliminated with the slag; but, if the ore is reduced as completely as possible, the greater part of the phosphorus goes with it, and when the ore is high in phosphorus the amounts of carbon and phosphorus in the product grow side by side. At Wärtsilä, ore containing 0.75 per cent. phosphorus was brought down to 0.10 per cent. in the raw loupe. Puddle-slags containing 4 per cent. phosphorus were brought down to 0.7 per cent. in the raw loupe in the same way. The charcoal used at Dobriansky is very inferior, being made mostly from Siberian fir, which produces a porous, friable charcoal, hardly capable of sustaining the weight of the charges. It is usually made in meilers, but badly done, and contains a large amount of moisture. The charcoal used at Wärtsilä is better, and seems to contain less moisture. The charcoal is assorted into two sizes, and is charged by weight, the coarse first, and then the fine, to fill up the spaces so that the ore should not pass through too rapidly. At Dobriansky some ores were roasted and others were not, but in either case they were crushed and passed through $\frac{1}{2}$- and $\frac{1}{4}$-inch mesh sieves; the roll-scale is used without any preparation. More recently, however, both ores, and the rich slag sometimes used, are crushed to quite a fine condition, with a corresponding improvement in the output. The chemical composition of the Malo-blagodatj magnetic iron-ores, according to five analyses is given below:

Analyses of Russian Magnetic Iron Ores.

	I.	II.	III.	IV.	V.
Iron	.....	.....	.....	.....	57.92
Fe_2O_3	65.01	63.01	69.1	52.34	.....
FeO	20.95	21.06	18.98	24.89	.....
MnO	2.79	3.64	2.84	2.25	2.64
MgO	2.62	3.77	2.25	1.2	3.67
CaO	1.10	1.58	1.12	1.2	2.83
Al_2O_3	0.42	3.00	0.56	2.7	3.06
SiO_2	5.41	4.43	5.2	11.9	7.09
Volatile Matter	0.42	.....	0.02	3.0	.....
Sulphur	.....	0.20	0.032	.....	.....
Phosphorus	0.15	0.12	0.06	.....	0.06
	98.87	100.63	100.342	98.48	.....

Analysis of Ores from Kuoinsk, on the Estate of Count Stroganoff, in the Government of Viatka.

	I.	II.
Iron,	48.14	50.30
MnO,	2.53	2.39
CaO,	4.32	3.92
Al_2O_3,	4.58	3.98
MnO,	0.55	0.80
SiO_2,	14.18	12.48

Analyses of Roll-Scale.

Iron,	66.00
SiO_2,	5.02

The following analyses, selected from a series of twenty-two, show the amounts of carbon, sulphur and silicon contained in blooms made by the Husgafvel furnace.

I. Made from a magnetic ore. Analysis after shingling and before reheating:

	Center.	Side.
Carbon,	0.07	0.06
Sulphur,	0.005	0.03
Silicon,	trace.	trace.

II. Made from Kuoinsk ore. Analysis after shingling and before reheating:

	Center.	Side.
Carbon,	0.13	0.07
Sulphur,	0.03	0.018
Silicon,	0.22	0.32

III. Made from roll-scale. Analysis after shingling and before reheating:

	Center.	Side.
Carbon,	0.07	0.32
Sulphur,	0.015	0.015
Silicon,	0.023	0.061

These analyses show a remarkably low degree of carbonization. It is, however, not considered best to have the iron so low in carbon, for it then has a tendency to absorb oxygen, which has a detrimental effect, similar to sulphur, causing the plates rolled from such blooms to crack along the edges. Blooms higher in carbon are consequently better; besides, the product is increased as the loss of iron in the slag is checked. The loupe should always contain a considerable amount of slag before it is shingled, as this prevents oxidation, and the metal from becoming "dry." For the sake of comparison, we give below the analyses of two puddled iron blooms, and one made of pig-iron, refined in a "Lancashire hearth or fire," all made of the same materials, and in the same iron-works at Dobriansky as the Husgafvel blooms.

PUDDLED IRON BLOOMS.

I.	End.	Middle.	End.
Carbon,	0.47	0.26	0.10
Silicon,	0.095		
Manganese,	0.04		

II.	End.	Middle.
Carbon,	0.05	0.07
Silicon,	——	——

LANCASHIRE HEARTH BLOOMS.

	End.	Middle.	End.
Carbon,	0.46	0.12	0.10
Sulphur,	0.01		
Silicon,	0.062		

The following table, showing the production of the Husgafvel furnace, was taken from one of the most recent official reports of the managers of the Dobriansky iron-works.

Average Production of:	With Unroasted and Finely Ground Magnetic Iron Ores.		With Roll-Scale.	
	Production of Steely Iron.		Production of Soft Iron Blooms.	Production of Steely Blooms.
	Charcoal of Fir.	Charcoal of Mixed Fir and Birch.	Charcoal of Fir.	Charcoal of Fir.
Shingled blooms in 24 hours.....	*Net Tons. 2.46	3 03	2.71	2.73
One bus'l charc'l produced—blooms	*Pounds. 8.00	12.52	11.03	11.52
†Bus'ls charc'l to make one ton blooms	250	159	182	174
The charges consisted of:				
Charcoal in Russian pounds.....	100	140	100	100
Ore and fluxes "	140–165	200–220	180–185	165–170
Number of charges in 24 hours.....	81.3	71.84	71.04	68.74
*Weight of each bloom in English lbs.	670	789	670	795
‡Pressure of blast in inches:	½–1⅓	⅔–1½	1½	¾–1¼
Temperature of blast (Centigrade)...	207°	230°	225°–230°	150°–230°
Maximum temperature of blast.....	234°	300°		

The above blooms were rolled directly into muck-bar with one heat, and then rolled and hammered into "Russian plate" by the usual process. A careful comparison of this plate with that made of puddle-iron revealed little difference, if any, the Husgafvel iron was more ductile, owing, possibly to its containing less sulphur. As far as I could judge, the loss from defective plate was about the same in both cases.

An interesting comparison might be made between the above results with those obtained by several charcoal blast-furnaces in the central districts of the Ural, the materials used being about the same in both cases.§

I. Cold-blast furnace. Charcoal, ⅓ birch and ⅔ fir. Production in twenty-four hours, 614 poods (10.23 tons) pig-iron. One bushel of charcoal produced 9.18 Russian pounds of pig-iron.

II. Hot-blast, 210° C. Same charcoal. Production in twenty-four hours, 883 poods (14.72 tons) pig-iron. One bushel of charcoal produced 13 Russian pounds of pig-iron.

* Original in Russian poods.

† Original in Russian korobs, one of which equals about 43 English bushels.

‡ Original in lines of mercury, one of which equals about $\frac{1}{12}$ of an inch.

§ See "Production of the Mines and Metallurgical Works of Russia," for the year 1884, by C. Kaulibin, St. Petersburg, 1886.

III. Cold blast. Same charcoal. Production in twenty-four hours, 606 poods (10.1 tons) pig-iron. One bushel of charcoal produced 9.63 Russian pounds of pig-iron.

From the best information I was able to obtain, the average prices of charcoal, ore and labor in the central districts of the Ural are about as follows:

Charcoal, per bushel,	3 to 6 cents
Ore,	$1.50 to $2.50
Fluxes,	$1.00 to $1.25
Wages, per day, from,	25 to 50 cents

Estimating that to make a ton of blooms under the conditions, present in the Ural requires on an average 190 bushels of charcoal, 2 tons of ore, and $\frac{1}{10}$ ton of fluxes, we have the entire cost, exclusive of interest on plant, etc., about as follows:

190 bushels of charcoal at 4½ cents,	$8.55
2 tons of ore at $2.00,	4.00
$\frac{1}{10}$ ton of fluxes, at $1.12,	.11
Wages at 40 cents,	1.60
Total,	$14.26

The repairs to this furnace are usually very slight. The official reports state that from the time the furnace was started in June, 1886, up to September 17th, 1887, the only repairs required were the replacement of a few iron parts of the movable hearths, whose weights were in the ratio of 0.017 poods for each pood of blooms produced.

The newest and largest Husgafvel furnace was erected in the spring of 1887 at the Russian government works at Petrozavodsk, on Lake Onega. It was started on August 14th, 1887, and I believe has been in almost constant operation ever since. The ores used there are lake- and bog-ores, very similar to those used at Wärtsilä, and the product, as at that place, is intended for use in the Siemens-Martin furnace with basic linings. In an official certificate in my possession, the following results obtained from this furnace are given:

Malleable iron obtained from the ore.	26.61 per cent.
One korob* charcoal produced,	11.4 pounds.
[One bushel charcoal produced	0.265 poods or 9.54 pounds].

* One korob equals 43 English bushels.

An ordinary blast-furnace, making charcoal pig-iron at the same works, and running on the same ores, etc., produced:

Pig-iron from the ore, . . .	30.29 per cent.
One korob charcoal produced, .	11.3 poods.
[One bushel charcoal produced, .	0.263 poods or 9.46 pounds].

A METHOD FOR THE ESTIMATION OF MANGANESE IN STEEL.

BY FRANK JULIAN, CHICAGO, ILL.

(Boston Meeting, February, 1888.)

THE determination of manganese by precipitation with potassium chlorate from a solution in concentrated nitric acid, filtration through asbestos, and solution in a reducing agent whose excess is estimated, is open to the objections that the filtration and subsequent solution of the binoxide are apt to be very slow, and that the asbestos may (according to Troilius), unless perfectly purified, affect the titration. In the following modification, filtration is avoided. I give the method as I use it for Bessemer steel.

In an 8-ounce Griffen beaker dissolve 1 gramme drillings in 15 c.c. nitric acid (1.2 sp. gr.). Evaporate to 5 c.c.; add 20 c.c. concentrated nitric acid and precipitate with chlorate as usual, avoiding a large excess; then add about 60 c.c. warm water, and 10 c.c. of the oxalic solution; stir until of a clear light yellow color, and titrate with permanganate while the solution is at about 80° C.

The binoxide should dissolve immediately when the oxalate is added. The titration is rapid, and the end-reaction, though transient, easily caught after a little practice. The standard solutions used are: 15 grammes crystallized ammonium oxalate to a liter, and 1.6 grammes potassium permanganate to a liter.

For standardizing the permanganate, to 10 c.c. of the oxalic solution is added 50 c.c. of hot water, and a solution of 1 gramme of steel in concentrated nitric acid, from which the manganese has been removed by potassium chlorate and filtration through asbestos.

I have made a large number of determinations in steel, pig-iron,

and ore with very satisfactory results for technical work, these being generally from .02 to .05 per cent. lower than by standard gravimetric methods; and the rapidity with which results can be obtained is greater than in any other method with which I am acquainted.

COAL-PRODUCTION IN UTAH, 1886.

BY CHARLES A. ASHBURNER, PITTSBURGH, PA.

(Utah and Montana Meeting, July, 1887.)

A VERY limited development has been made of the coal-fields of this Territory. This is to be accounted for by the fact that a number of the coal-fields lack railroad transportation, by the distance of other fields from the consumers (due to restricted settlements), and finally by the poorer quality of a number of coals which have been mined as compared to others which can be economically imported into the Territory.

The consumption of coal in Utah during 1886, outside of that used by the railway companies, was:

	Tons.
Utah coal,	110,000
Wyoming coal,	19,000
Colorado coal,	4,945
Total,	133,945

It is estimated that there are 1100 miles of railroads in the Territory which use coal for fuel. The consumption of these roads for the year is estimated at 100,000 tons, making an aggregate consumption in the Territory of 223,945 tons. The consumption of coal in Salt Lake City during the year was as follows:

	Tons.
From Grass Creek mines,	21,000
" Pleasant Valley Company's mines,	12,700
" Utah Central Company's mines,	9,000
" Rock Spring mines,	6,500
" Almy mines,	5,000
" Miscellaneous sources,	5,800
Total,	60,000

Besides the coal consumed in Utah, there was consumed during 1886 24,793 tons of coke which was brought in from Colorado, and 9,090 tons from other localities.

The first coal operation started in the Territory was in the vicinity of Coalville, in Summit County, northwest of Salt Lake City, at the mines which are now operated by the Home Coal Company. These mines were opened in 1884. Prior to this there were a number of local coal openings in the Territory, but the product of none of them was sufficient to make the mines of commercial importance. The mines at present operated for shipment to the trade are confined to the counties of Summit, San Pete and Emery.

The Home Coal Company operates conjointly what are known as the Wasatch and Crimson mines in the vicinity of Coalville. These mines, during 1886, produced in the aggregate 24,417 tons of merchantable coal, which was shipped principally to Park City and Salt Lake City.

Very little coal is sold at the mines, and the fuel used at the mines consists of screenings from the refuse slack-coal. During the year the average selling-price on cars, delivered at the mouth of the mine, was $2.25 per ton. The bed is about twelve feet thick and is composed rather of lignite than of coal. It is principally adapted for use as a domestic fuel and for stationary boilers.

The company employs about twenty-five miners, who are paid ninety cents a ton for breaking the coal and loading it in the mine-cars. The ton used contains 2150 pounds. The average wages of laborers other than these miners is $1.85 a day; two hundred and fifty days were worked during the year.

The Grass Creek mines, in the vicinity of Coalville, produced during the year 29,131 tons, 26,295 tons of which were shipped to outside markets and 436 tons consumed at the mines. The average thickness of the bed at these mines is ten feet, and the average number of miners employed daily during the year was twenty-four.

The Winter Quarter mines of the Pleasant Valley Coal Company are situated at Scofield, in Emery County. The product of this company during 1886 was 71,814 tons (2000 pounds), being greater than that of any other one county. The coal is shipped principally to points in Utah, to Western Colorado, and to Butte, in Montana.

The average selling-price of the coal delivered on cars at the mouth of the mines during the year was $2.05. The thickness of the bed is 11 feet, and the following is an analysis of the coal, furnished by Mr. W. F. Colton, Secretary of the Company:

	Per cent.
Water,	6.55
Volatile matter,	39.75
Fixed carbon,	47.65
Ash,	6.05
	100.00

This coal, or rather lignite, is supposed to be a superior steam fuel.

The mine of the Utah Central Railroad Company is also located at Scofield, in Emery County. The total product of this mine during the year was 34,676 tons (2000 pounds), 500 of which were consumed at the mine, and the remainder sold at an average price of $2.00 per ton at the mouth of the mine, for shipment to Salt Lake City, Provo, and Ogden. The bed is 28 feet thick, without any partings of rock, clay or shale. About 25 feet of the bed is mined, 3 feet being left as roof. The Company employed about 60 miners during the winter months, and about 30 during the summer months; wages for the miners being 70 cents per ton.

An analysis of the upper part of the bed which is left in the mine, made by Mr. A. S. McCreath, showed as follows:

	Per cent.
Water,	5.798
Volatile matter,	40.487
Fixed carbon,	51.950
Ash,	1.765
	100.000

Mr. W. D. Sharp, Superintendent of the coal department of the Utah Central Railway, reports to me that the top layer of coal which is left in the mine is much brighter than the coal below it, and is freer from the white spots (probably sulphate of lime) which were found in other parts of the bed.

The Central Pacific Coal & Coke Company, Limited, owns about 10,000 acres of coal lands in San Pete County, which produced during the year about 2000 tons of coal, which was consumed entirely by the San Pete Valley Railway. The bed is from 6 to 8 feet thick, and is locally considered a semi-bituminous coal. It is asserted that this coal is the only coal hitherto mined in Utah which will make a desirable coke. The miners receive $1.90 per ton for mining, screening and loading on the cars.

The total product of the territory including that given above, and that of small mines from which no returns have been obtained, is estimated at 200,000 tons.

Table Giving the Production by Companies.

	Tons.
Pleasant Valley Coal Company,	71,814
Utah Central Coal Company,	34,676
San Pete Coal & Coke Company,	1,800
Grass Creek Coal Company,	29,131
Home Coal Company,	24,417
Small mines unreported,	38,162
Total,	200,000

The following are the statistics of the Coal Department of the Union Pacific Railway for 1886, showing special facts connected with the Grass Creek mines in Utah and three mines in Wyoming.

	Wyoming.			Utah.
	Carbon.	Rock Springs.	Almy.	Grass Creek.
Tons produced	214,235	359,238	156,618	29,131
Delivered to R. R. Co	178,904	128,672	116,067	24,016
Shipped to outside markets	33,735	228,361	41,602	26,294
Consumed at mines	1,596	2,205	604	436
Average monthly No. of miners.	223	390	244	24
Principal outside market	Nebraska and Kansas.	All points reached by U. P. Ry.	Montana and Utah.	Utah.
Thickness of coal-bed	7 feet.	5 to 12 feet.	18 feet.	10 feet.

The outlook for the further development of the Utah coal-fields is promising, and the production of the mines during 1887 will no doubt be much greater than it was for 1886. Exaggerated statements of the annual production of coal in the Territory have been circulated in various quarters during the past few years, and any inconsistencies between these statements and the figures here given is no evidence that the total annual production has really fallen off; on the contrary, it has increased from year to year.

THE CHLORINATION OF GOLD-BEARING SULPHIDES.

BY E. GYBBON SPILSBURY, NEW YORK CITY.

(Utah and Montana Meeting, July, 1887).

In a paper* read at the last meeting of the Institute, in Scranton, Pa., I referred to the proposed introduction at the Haile gold-mine, South Carolina, of a novel method of chlorination. I have since

* *Trans.*, xvi., 767.

then been experimenting with this method, and operating it on a regular scale; and I think it may be of interest to members to have a few figures showing the results obtained.

The system, which is known as the "barrel system," has been brought to its present state of perfection by our fellow-member, Mr. Adolph Thies, to whom all the credit for the improvement should be given. In reality it is an outcome of the Mears process. The latter process, as is well known, consists in revolving the charge of ore to be chlorinated in a lead-lined barrel, supported on hollow trunnions, through which the chlorine, generated by the ordinary method, is forced under pressure. This method, while fairly effective metallurgically, is very objectionable in some particulars, especially the difficulty of keeping tight joints in the stuffing-boxes and the consequent escape of chlorine gas through the works. The cost of repairs has also proved excessive.

The system I have now adopted is extremely simple, and requires no tight joints liable to leakage. The chlorinator consists of a plain iron barrel lined with lead, and provided with a man-hole on one side for charging and discharging the ore. The ore is charged in one-ton lots. The barrel is first partially filled with water; then a sufficient quantity of chloride of lime is dropped in, on top of which the roasted ore is charged. On top of the ore is poured the requisite amount of sulphuric acid; the man-hole cover is put in place and thoroughly secured, and the barrel is started to revolve. The reason for charging in this order is to prevent the sulphuric acid and chloride of lime from coming in contact before the barrel has been securely closed. When the charging is completed the barrel is started and is kept revolving until the whole of the gold is dissolved. A very ingenious arrangement has been devised which enables the operator, by means of a clay pop-valve, to test from time to time, without opening the barrel, the presence of an excess of chlorine gas. With the sulphides I operated on, each one-ton charge consumed 20 pounds of chloride of lime and 25 pounds of acid. The time occupied in chlorinating so far has been six hours, but as we gain experience I expect to be able to reduce this time by nearly one-half.

When this operation is completed, enough water is introduced to nearly fill the barrel, which is again revolved, so as to thoroughly wash the ore and dissolve the chloride of gold. The barrel is then opened and the liquor is decanted off upon large shallow filter-beds. Another wash-water is then put into the barrel, which is again

revolved for a few minutes and then decanted upon the filter. Ultimately the whole charge is turned down in the ordinary way, and a final wash-water is given on the filter. The results so far obtained have been remarkably successful. I append herewith the result of the last eight charges run. Each charge contained a little over one ton of ore of such fineness that 99 per cent. of it would go through a 100-mesh, and 60 per cent. through a 150-mesh screen. The assays made of the roasted ore showed 36.70 gold per ton.

The assay of the tailings from			first charge	gave	$1.56	per ton.
"	"	"	second	"	1.03	"
"	"	"	third	"	0.78	"
"	"	"	fourth	"	0.52	"
"	"	"	fifth	"	trace	"
"	"	"	sixth	"	0.78	"
"	"	"	seventh	"	trace	"
"	"	"	eighth	"	0.78	"

In order to further test the advantages of this system, I tried one charge considerably heavier than the others, and endeavored to filter it direct in the usual way. While I used for this charge nearly double the amount of wash-water, and while the time occupied in filtering was much more than the other charges required, the tailings resulting contained $5.68 per ton. There seems to be no difficulty in filtering through a bed of ore from 3 to 4½ inches thick, but when the thickness of the bed is greater, then the filtering of such very fine material becomes almost impossible; hence the advantage of decanting.

The precipitation is done in the usual manner with proto-sulphate of iron. Our ores contain a considerable percentage of magnesia, which at first gave us a very voluminous precipitate. I find, however, that the precipitation of lime and magnesia may be entirely prevented if care is taken to keep the tank-solution slightly acidulated. The moment the solution is allowed to become neutral the magnesia begins to show in the precipitating tank as a milky cloud, which, however, immediately disappears on the addition of a small proportion of free acid.

The cost of roasting, labor, chemicals, and power with us will never exceed $4.00 per ton. So far it has not attained that figure, some of my work having been done as low as $3.15 per ton. The actual gold recovered is in all cases within a few cents per ton of the full assay-value of the ore, less the amount shown by assay to have

remained in the tailings. While the gold obtained in our stamp-mill and amalgamating works is seldom purer than 897 fine, the gold obtained by this process is generally over 978 fine.

THE RUSSELL PROCESS IN ITS PRACTICAL APPLICATION AND ECONOMIC RESULTS.

COMPILED FROM MR. RUSSELL'S NOTES BY ELLSWORTH DAGGETT, SALT LAKE CITY, UTAH.

(Boston Meeting, February, 1888.)

INTRODUCTION.

In the first paper on the Russell process presented by Mr. Stetefeldt, in May, 1884 (*Transactions*, xiii.), the process was treated from a purely theoretical standpoint. In his second paper of October, 1886 (*Transactions*, xv.), the details of the plant, the chemicals used, and the manipulation in both mill and laboratory were treated. The aim of the present paper is to present economical results recently attained, new data affecting the preparation and manipulation of ore in the mill, and a more complete and systematic scheme of laboratory work.

This paper is compiled mainly from Mr. Russell's notes of his own experience. Many of these notes, particularly those relating to work at Cusihuiriachic,* Mexico, have been upon work directly under the personal observation of the compiler, who takes pleasure in thus presenting to the Institute, data and considerations of much interest and value not only to users of the Russell process, but to mill men generally.

The description of the chemicals used and of the preparation of the mill-solutions being in part the same now as when Mr. Stetefeldt's second paper was published, some use is here made of portions of that paper.

No theoretical matter is here given. For a treatise containing also the theory of the process the reader is referred to Mr. Stetefeldt's complete work on "Lixiviation of Silver Ores by Hyposulphite Solutions," now in preparation.

To facilitate the consultation of this paper, the following synopsis and index is prefixed. The figures in parenthesis indicate the page of this paper on which the subject is treated.

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* Called "for short," in this paper, as in common usage among mining men, "Cusi."

I. Laboratory Work.

A.—Leaching-tests by extra-solution or Russell process (368).
a. Leaching-tests for raw ores and for tailings (369).
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I. Laboratory Work.

A.—*Leaching-Tests by Extra-Solution or Russell Process.*

Different ores require different treatment on the small scale, in the assay-office, as well as in the mill. Consequently, for the reasons explained further on, eight different leaching-tests are used in testing raw ores, and eight for roasted or chloridized ores. Of the eight methods, the one giving the highest per cent. extraction is then selected for all further leaching-tests on that particular ore.

In making the tests the following points should be noticed:

The metric system and assay-ton weights are supposed to be used. The weight of ore taken is $\frac{1}{3}$ or $\frac{1}{2}$ an assay-ton, and is placed in a beaker of about 300 c.c. capacity.

If the fire-assays are made by scorification, of course a smaller amount of ore should be taken, but no change need be made in the amount of chemicals or solution used. Also, if beakers of 300 c.c. are not at hand, smaller sizes and less volume of solution can be used, without altering the weight of chemicals. The given proportion between hyposulphite and blue-stone should be observed.

The preliminary washing with water in the case of roasted or chloridized ores is by decantation, the clear liquid being decanted upon the filter that is afterwards used to receive the ore.

The amount of hyposulphite of soda used is 20 grammes, unless otherwise stated.

The "blue-stone solution" referred to is a 20 per cent. solution made by dissolving 200 grammes blue-stone in hot water and diluting to 1,000 c.c. Unless otherwise stated, 25 c.c. of this solution is used.

In all cases the contents of the beakers should be occasionally stirred.

The washing with water at the end of the operation is to remove the last traces not only of dissolved silver, but also of hyposulphite of soda, which would interfere with the burning of the filter. After the final washing with water, the filter and ore are transferred to a flat dish and dried. When dry, the ore is separated from the filter and the filter burned—the ash being added to the ore, which is then ready to be assayed. If the fire-assays are made by scorification, the presence of the filter-ash is inconvenient. For such assays, the last traces of the ore are removed from the dry filter with a stiff brush, and the filter-paper is not assayed.

The following are the methods of making lixiviation-tests by the

extra-solution (Russell process) in the assay-office. The first eight are for raw ores and tailings—the second eight for roasted or chloridized ores. For brevity the terms "Hypo" and "Blue" are here used for the salt hyposulphite of soda and the blue-stone solution respectively.

a. Leaching-Tests for Raw Ores and for Tailings.—1. Add 250 c. c. cold water and 20 grm. Hypo; let stand 12 to 16 hours cold; decant; add 25 c.c. Blue; dilute to 300 c.c. with cold water; let stand 12 to 16 hours cold; add 20 grm. Hypo; let stand 12 to 16 hours cold; filter and wash.

2. Add 250 c.c. cold water and 20 grm. Hypo; let stand 12 to 16 hours cold; add 20 grm. Hypo and 25 c.c. Blue; let stand 12 to 16 hours cold; filter and wash.

3. Add 100 c.c. cold water and 25 c.c. Blue; let stand 12 to 16 hours cold; dilute to 300 c.c. with cold water; add 20 grm. Hypo; let stand 12 to 16 hours cold; filter and wash on filter with hot Hypo solution.

4. Add 100 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 200 c.c. cold water and 20 grm. Hypo; heat to about 130° F.; filter and wash.

5. Add 250 c.c. cold water and 25 c.c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat on sand-bath to about 130° F.; decant; add 200 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat, filter and wash.

6. Add 200 c.c. cold water and 90 c.c. Blue; let stand one-half hour cold; add 90 grm. Hypo; heat to about 150° F. and wash.

7. Add 250 c.c. cold water and 25 c.c. Blue; heat to about 130° F.; decant; add 20 grm. Hypo and 25 c.c. Blue; dilute to 300 c. c. with cold water; heat to about 130 ° F.; filter and wash.

8. Add 200 c. c. hot water and 90 c. c. Blue; let stand one-half hour; add 90 grm. Hypo; heat to about 150° F.; filter and wash.

b. Leaching-Tests for Roasted or Chloridized Ores, Mattes and Tailings.—1. No washing; add 50 c.c. cold water and 25 c.c. Blue; let stand 2 hours cold; dilute to 300 c. c. with cold water; add 20 grm. Hypo; let stand 2 hours cold; filter and wash.

2. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 12 to 18 hours cold; add 20 grm. Hypo; let stand 12 to 18 hours cold; filter and wash.

3. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 2 hours cold; add 20 grm. Hypo; let stand 2 hours cold; filter and wash.

4. Wash with cold water; add 25 c. c. Blue; dilute to 300 c. c. with cold water; let stand 12 to 18 hours cold; add 20 grm. Hypo; let stand 12 to 18 hours cold; filter and wash on filter with hot Hypo solution.

5. No washing; add 250 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat to about 130° F.; decant; add 250 c. c. cold water and 25 c. c. Blue; let stand 1 hour cold; add 20 grm. Hypo; heat to about 130° F.; filter and wash.

6. Wash with hot water; add 200 c. c. hot water and 90 c. c. Blue; let stand one-half hour; add 90 grm. Hypo; heat to about 150° F.; filter and wash.

7. Wash with hot water; add 90 c. c. Blue; let stand one-half hour; add 90 grm. Hypo; dilute to 300 c. c.; heat to about 150° F.; filter and wash.

8. Wash with hot water; add 20 grm. Hypo; let stand (with residue of wash water, about 30 to 40 c. c.) 1 hour; add 25 c. c. Blue; dilute to 300 c. c. with cold water; heat to about 130° F.; filter and wash.

B.—*Leaching-Tests by Ordinary Solution.*

In the treatment of roasted ores and sometimes of raw ores, assay-office leaching-tests by the "ordinary" solution also, as well as by the "extra," should frequently be made. If these tests are made by the mill-solution, as is done in some mills, the results may be very misleading. For instance, if caustic soda or lime were present in the mill-solution, such a solution on many ores would give very low results, both in assay-office and mill. The low extraction would then be laid to a change in the character of the ore or to the furnace-work, whereas the true cause of the difficulty would be the impurity in the solution. By the use of a fresh hyposulphite solution in the assay-office test, the cause of the low results in the mill would become immediately apparent.

In the case of roasted or chloridized ores which have been so coarsely crushed in the mill as to require for assaying a re-crushing in the assay-office, a metal mortar should never be used, as may be seen from Table I. As there shown, the per cent. of extraction by the ordinary solution in the assay-office, on Cusi ore, is reduced 10 to 11 per cent. by the use of the iron mortar, while the results by the extra-solution are practically unaltered. The cause of this is probably the reduction of part of the silver to the metallic state, in which state it is much more soluble in the extra-solution than in

the ordinary. But on the Lake Valley ores, the use of the iron mortar reduced not only the extraction by the ordinary solution to

TABLE I.

EXPERIMENTS ON THE RESULTS OF GRINDING ROASTED ORE IN WEDGEWOOD OR AGATE, AND IN IRON MORTARS.

Name of Ore.	Mesh of Screen and Method of Crushing.	Kind of Sample.	Material of Mortar.	Mesh of Screen Used in Assay Office.	Value of Ore.	Value of Tailings from Ordinary Solution.	Value of Tailings from Extra-Solution.	Per cent. extracted by Ordinary.	Per cent. by Extra.
					Ounces per ton.				
Cusi.	20-Screen. Stamps.	Sample from Furnace	Wedgewood.	40	30.2	6.4	2.8	78.9	90.7
		Sample from Furnace	Iron.	40	30.2	9.6	2.8	68.3	90.7
		Sample of Tub Charges	Agate.	40	30.0	4.4	2.6	85.3	91.3
		Sample of Tub Charges	Iron.	40	30.0	7.4	2.8	75.4	90.8
Lake Valley.	16-Screen. Rolls.	Sample from Furnace	Agate.	40	16.0	2.9	2.0	81.3	87.5
		Sample from Furnace	Iron.	40	16.0	6.7	6.0	58.1	62.5
		Sample from Floor	Agate.	40	18.0	3.7	3.5	79.4	80.6
		" " "	Iron.	40	18.0	7.2	6.0	60.0	66.7
		Sample of Tub Charges	Agate.	40	18.0	3.0	3.0	83.3	83.3
		Sample of Tub Charges	Iron.	40	18.0	9.0	5.7	50.0	68.3

the extent of 25.3 per cent., but also the extraction by the extra-solution 18 per cent. No explanation can be given of the change

TABLE II.

EFFECT OF USING A COPPER VESSEL ON THE LEACHING-TEST BY THE ORDINARY SOLUTION IN THE ASSAY-OFFICE.

Roasted Ore.	Value in Ounces per Ton.		Ounces in Favor of Copper Vessel.	Per Cent. in Favor of Copper Vessel.
	Tailings from Solution heated in Copper Vessel.	Tailings from Solution heated in Wooden Vessel.		
78 80	20.52	24.84	4.32	5.8
74.76	8.88	10.32	1.44	2.0
77.46	10.92	15.24	4.32	5.6
60.00	13.80	20.16	6.36	10.4
72.00	15.30	19.56	4.26	5.9
75.60	9.84	11.52	1.68	2.2
76.52	9·84	12.90	3.06	4.1
63.96	14.94	16.82	1.88	3.0
69.30	15.96	18.60	2.64	3.8
72.78	8.40	9.12	1.02	5.4
72.72	8.28	10.98	2.11	3.7
71.70	12.42	15.46	3.0	4.7

taking place in this case. On some roasted ores the percentage of extraction by the ordinary solution in the assay-office is reduced 50 to 60 per cent. by the use of a metal mortar.

As observed by Mr. Frank Johnson, of the Yedras mill, in making leaching-tests by the ordinary solution, the solution should never be heated in a copper vessel. Table II. shows the difference, in the results by the ordinary leaching-tests in the assay-office, caused by heating the ordinary solution in a copper vessel, as compared with a glass or wooden one. This difference is caused by the dissolving of a very small portion of the copper vessel by the hot ordinary solution, thus making a very weak extra-solution.

C.—*The Best Assay-Office Leaching-Methods by the Extra-Solution for Various Roasted Ores, and the Results Obtained.*

Table III. gives the results obtained by assay-office leaching-tests by extra-solution on average samples of 18 roasted ores, of which 4 were roasted in Stetefeldt, six in Howell, three in reverberatory, and 5 in Brückner furnaces. The table gives also for each ore the two best methods of leaching-tests by extra-solution in the assay-office.

TABLE III.

METHODS USED, AND RESULTS OF LEACHING-TESTS BY EXTRA-SOLUTION, IN ASSAY-OFFICE, ON ORES ROASTED IN STETEFELDT, HOWELL, REVERBERATORY AND BRÜCKNER FURNACES.

Name of Mine.	How Crushed.	Mesh of Screen.	Per cent. of Salt Used.	Kind of Furnace.	Per cent. of Silver Extracted.		Leaching Methods Most Applicable.
Ontario, Park City	Stamps.	16	12	Stetefeldt.	97.1	95.5 Per ct.	6 or 7
Daly, " "	"	26	11	"	95.2		2 or 1
Lexington, Butte	"	30	15	"	93.2		8 or 5
Manhattan	"	40	6	"	96.7		1 or 3
Alice, Butte	"	26	15	Howell.	93.5		7 or 1
Sierra Grande, Lake Valley	Rolls.	16	7	"	84.6	91.3 Per cent.	5 or 7
Pyramid	Stamps.	30	.....	"	93.6		7 or 6
San Antonio, Chihuahua	"	12	10	"	91.0		1 or 8
San Miguel, Chihuahua	"	12	8	"	93.6		1 or 3
Chloride, New Mexico	Rolls.	30	7	"	91.6		
Yedras, Sinaloa	Stamps.	26	7	Reverberatory.	90.5	90.7 Per ct.	4 or 2
Sombrerete, Zacatecas	Rolls.	20	10	"	94.9		5 or 3
Carmen, "	"	20	0	"	86.7		3 or 4
Veta Grande, Chihuahua	Stamps.	16	6	Brückner.	87.8		3 or 1
Custer, Idaho	"	30	5	"	95.1	89.6 Per cent.	1 or 2
Bertrand, Nevada	Rolls.	12	6	"	83.9		5 or 8
Tybo, Nevada	Stamps.	30	.....	"	93.1		5 or 1
North Mexican, Chihuahua	"	30	.....	"	82.9		6 or 7

The reasons for employing eight different methods in the first testing of an ore are as follows:

1st. The assay-office extraction, which serves as a standard for mill-work, should always be the highest attainable by any use of hyposulphite and blue-stone.

2d. Collecting all the results by various methods on many samples of roasted ores, it is found that in the eight methods given are comprised all those which give the highest results on one or more samples of the ore investigated—these samples representing the ores treated in the principal mills of the United States and Mexico.

3d. If the best one of only two or three methods is used in the assay-office, the extraction in the mill may be greater than in the assay-office. Examples of this may be seen in Table IV., in the cases of Ontario, Lake Valley and Sombrerete ores. At Lake Valley (and also at the Rubio mill in Parral) the extraction in the mill was sometimes 6 to 10 per cent. above the extraction in the assay-office.

TABLE IV.

COMPARISON BETWEEN THE RESULTS BY RUSSELL PROCESS IN THE ASSAY-OFFICE AND THE RESULTS BY RUSSELL PROCESS IN THE MILL.

Name of Mine.	Kind of Furnace Used.	Mesh of Screen.	Per cent. of Salt.	No. of Assay Office Extra Solution Leaching Methods Tried.	Per cent. by Best one of those Methods in the Assy Office.	Mill Per cent.	Per cent Extracted in Assy Office more than in Mill.
Ontario	Stetefeldt.	30	0	2	85.8	85.7	+ 0.1
"	"	30	9½	2	94.7	96.2	— 1.5
"	"	20	12½	2	97.0	97.0	0.0
"	"	16	7½	2	89.0	89.5	— 0.5
"	"	16	8	2	91.5	92.0	— 0.5
"	"	16	12	2	96.3	97.1	— 0.8
"	"	16	16	2	95.0	95.7	— 0.7
"	"	16	18	2	95.9	95.1	+ 0.8
Lake Valley	Howell.	16	7	2	82.6	85.3	— 2.7
Sombrerete	Reverb'y.	8 screen but with fine half out.	10	2	86.2	86.5	— 0.3
"	"	8	10	2	90.6	91.3	— 0.7
San Bartolo	Howell.	16	9	8	88.5	87.3	+ 1.2
San Antonio	Howell.	26	10	8	89.2	88.5	+ 0.7
"	"	12	10	8	90.3	83.9	+ 6.4
San Miguel	Howell.	26	8	8	91.9	90.4	+ 1.5
"	"	12	8	8	89.1	84.0	+ 5.1
Yedras	Brückner.	26	7	8	83.1	82.6	+ 0.5

4th. If the best one of only two or three methods is used in testing a new ore, the applicability of the process to that ore is not determined.

Usually there is not much difference in the results obtained by the eight methods; but sometimes, as in the case of the Yedras roasted ore, the difference is considerable. On that ore, a strong hot extra-solution in the assay-office, as in method 6 or 7, gave only 36 per

cent. extraction. The mill-results on the same ore with a strong hot solution were 86 per cent. If only methods 6 or 7 had been employed in the preliminary testing of Yedras ore, the process might have been supposed inapplicable to that ore. On that ore the highest percentage in the assay-office was obtained by a cold, weak extra-solution, the result of which agreed with the results of a strong hot extra-solution in the mill.

It should be observed that the terms "strong" and "weak" do not mean the same in the mill as in the assay-office—the weakest extra used in the assay-office being stronger than the strongest in the mill.

In general, the method most applicable in the assay-office is not the most applicable in the mill, and frequently, as in the case of the Yedras ore, they are directly opposite. As pointed out further on, the proper methods for mill-work on roasted ores are very accurately indicated by the alkalinity or acidity of the first wash-water. For all raw ores, the manipulations are simpler and the methods very much more uniform.

D.—*Determination of the Percentage of Soluble Salts in Roasted Ore.*

In calculating the apparent percentage of silver extracted in the mill from roasted ores, it is necessary to know the percentage of salts which have been extracted from it by the first wash-water and the leaching-solutions.

If the ore contains no hydrate or sulphate of lime, or other salts of about the same solubility in water, the determination of the solubility is a simple operation: 10 or 20 grammes of the ore is placed on a weighed filter and leached with 500 to 600 c. c. of hot water. The residue and filter are then weighed. This method will give the amount of soluble salts extracted from the ore in the mill, usually within one-half and nearly always within three-quarters of 1 per cent., the amount of gold, silver, copper and lead salts which are extracted by the regular leaching-solutions rarely exceeding one-half of 1 per cent.

But if the roasted ore contains hydrate or sulphate of lime, the above method gives too high a solubility, owing to the solubility of these salts in a large amount of water. For instance, at Yedras, the solubility as determined by the above method was over three times as great as it actually was in the mill. In such cases the determination of the solubility is as follows: 10 or 20 grammes of the

roasted ore is placed on a weighed filter and the same amount of the mill-tailings corresponding to that ore is placed on another weighed filter. Each is leached first with 300 to 500 c. c. of cold water and then with 2000 to 2500 c. c. of a strong (8 to 12 per cent.) hot hyposulphite solution. These volumes will be sufficient to dissolve all the salts which are in the least degree soluble in water or hyposulphite. The two residues and filters are then dried and weighed. The difference between the percentage of solubility of the ore and that of the tailings, gives, with sufficient exactness, the percentage of salts which has been extracted in the mill by both water and solutions. The slight error is due to the fact that the ore and tailings are not of the same nature—the latter having been concentrated previously to the weighing, by an amount equal to the solubility in the mill. But the determination is sufficiently exact for practical purposes, as an error of 1 per cent in the solubility makes very little difference in the apparent extraction, as shown in the following table, which also illustrates the mode of determining the apparent extraction. The two solubilities are assumed as being 9 and 10 per cent. respectively:

	Example I.	Example II.
Value of roasted ore, . . .	25 0 oz.	25.0 oz.
Percentage of solubility in assay-office,	9.	10.
Calculated value of ore, . . .	27.47 oz.	27.78 oz.
Value of mill-tailings, ...	2.5 oz.	2.5 oz.
Apparent mill-extraction, . . .	90.9 per cent.	91.0 per cent.

The determination of the percentage of solubility should be made each day, or if the leaching-tanks have a capacity of 20 to 50 tons, it should be determined for each charge of ore.

E.—*Determination of the Strength of the Hyposulphite or Stock-Solution.*

As it is important that the proper relation should be observed between the amount of hyposulphite and that of blue-stone in the extra-solution, the strength of the hyposulphite or stock-solution should be determined whenever the extra-solution is to be made up for a large charge of ore. For small charges, say of ten tons each, a determination of the strength for each charge would be inconvenient; hence the strength is tested only once each day.

The best method of determining the strength of the hyposulphite solution is by starch and iodine. The reactions for this method are given in works on chemistry, but a description of the practical application of this method to the leaching process cannot safely be

omitted here. If a small quantity of starch is present in a hyposulphite solution, and a solution of iodine is added, drop by drop, the sensitive blue color of iodide of starch will continue to be destroyed as fast as it is produced, as long as any hyposulphite exists. As soon as the hyposulphite has been converted into tethrathionate, the least excess of iodine will act upon the starch, and the blue color will be permanent. This method will give the total amount of hyposulphite, no matter in what form it exists, whether as sodium, calcium or silver hyposulphite, except as it exists in the extra-solution, or in the presence of caustic alkali. In the latter case, the method will indicate the presence of more hyposulphite than actually exists. Consequently, the hyposulphite solution used for the test should be neutralized (if alkaline), particularly in treating roasted lime ores, by the addition of a few drops of dilute sulphuric acid, before adding the iodine solution. No harm is done if the solution is slightly acid, provided the iodine and starch are immediately added. As just stated, the method is not applicable for the determination of the hyposulphite in the extra-solution, so that the sample of stock-solution must be taken from the solution which is to be used for the extra-solution before the blue-stone is added to it.

The standard iodine solution for a volume of one liter is made up as follows: The iodine is such as can be obtained at any drug store, no re-subliming being necessary. 5.11 grammes of the iodine is placed in a beaker of at least 200 or 300 c.c. capacity, which is set in a moderately warm place on the sand-bath at an angle of 20 or 30 degrees from the perpendicular. The iodine is then covered with about its own weight of iodide of potassium, and about 15 c.c. of water. In about twenty minutes, the iodine will be nearly or quite dissolved. About 200 c.c. of cold water is then added and the clear liquid decanted off. If any iodine still remains undissolved about three or four more grammes of iodide of potassium is added. As a little water has probably been left in the beaker from the decantation, no more need be used. The beaker is placed as before in an inclined position in a warm place. Usually this second addition of iodide of potassium will result in the complete solution of the iodine. If not, the second operation should be repeated. The whole solution is then diluted to 1000 c.c. with cold water and placed in a glass-stoppered bottle. 100 c.c. of this solution should correspond to one gramme of hyposulphite of soda ($Na_2S_2O_3 + 5aq$). But, as the solution may slightly decompose during the first few days after its preparation, it is best to use 5.5 grammes of iodine

instead of 5.11 grammes in making up the standard solution. After a few days, the solution may then, if necessary, be diluted by the addition of a small amount of water, so that 100 c.c. shall correspond to one gramme of hyposulphite. This exact proportion is only necessary for the convenience of the man in charge of the leaching in the mill, by whom the solution is used.

The preparation of the starch solution is as follows: About 10 grammes of common starch is boiled or heated in about 500 to 1000 c.c. of water. Enough salt is then added to this solution to completely saturate it. The clear or turbid liquid is then decanted into a stoppered bottle, leaving the undissolved salt behind. The object of the addition of salt is to preserve the starch solution, which would otherwise decompose, and have to be made fresh every few days.

To standardize the iodine solution, about .1 to .2 of a gramme of a clear crystal of commercial hyposulphite of soda is weighed out in the fine assay-balances used for weighing gold and silver buttons. It is placed in a beaker with about 20 c.c. of water and 2 to 4 drops of starch solution. The iodine solution is then run in from a burette, until the blue color of iodide of starch is permanent.

The determination of the strength of the stock-solution is made in the mill by the man in charge of the leaching at the time the extra-solution is to be prepared. The reagents required are the iodine solution, the starch solution, and, if the ore is alkaline roasted, some very dilute sulphuric acid. The apparatus consists of a 10 c.c. pipette, a 100 c.c. burette graduated to $\frac{1}{5}$ or $\frac{1}{10}$ of a c.c., a glass beaker or tumbler and a glass stirring-rod. These are kept in a small closet near the leaching-tanks. As each charge of ore weighs nearly the same, each charge of extra-solution is always of the same volume. Consequently, a table can be prepared showing the number of pounds of hyposulphite present in the solution, to which the blue-stone is to be added, corresponding to any percentage of hyposulphite as shown by the test. In making up the extra-solution therefore, the man in charge measures out 10 c.c. of the stock-solution, adds a few drops of starch solution, runs in the iodine solution from the burette and reads off the percentage of hyposulphite. By referring to the table, he finds opposite this percentage, the amount of hyposulphite, say 390 pounds present in the volume of stock-solution which is to be used for the preparation of the extra. As the amount of blue-stone for a 50 ton charge at the rate of, say, 5 pounds per ton, would be 250 pounds, and as the amount of

hyposulphite for the extra-solution must be twice as great as the weight of blue-stone, the amount of hyposulphite to be added with the blue-stone is 110 pounds.

F.—*Assaying of the Sulphides for Silver.*

The set of experiments shown in Table V. was made by Louis Janin, Jr., when in charge of the mill at Cusi. In experiments 1 and 2, of the 55 grammes of granulated lead, 15 were first placed in the scorifier and hollowed out to receive the sulphides, which should not touch the sides or bottom of the scorifier. The remaining 40 grammes were placed on the top of the sulphides.

In No. 4, the sulphides were first treated with nitric acid and the silver was precipitated from the filtrate with hydrochloric acid and filtered off—this filter, with its contents, then being added to the filter containing the residue undissolved by nitric acid. The two filters and their contents were then dried and assayed.

For assaying cupels, pulverize the cupel through a 30-mesh screen. Use 30 grammes fused borax, 30 grammes litharge and 30 grammes of carbonate of soda, and cover with salt. Use a slow fire until nearly fused. If the crucible tends to boil over, throw in 20 to 25 grammes of salt. For slags, all the slag from the assay is saved and pulverized through a 20-mesh screen, and mixed with flux used in experiment No. 4.

TABLE V.

EXPERIMENTS IN ASSAYING "SULPHIDES."

Number of Experiment.	Method of Assay Used.	Weight of Sulphides, A. T.	Weight of Fluxes Used.						Uncorrected Assay Value, oz. per ton.	Silver in Slag, Per cent.	Silver in Cupel, Per cent.	Total in Slag and Cupel, Per cent.	Corrected Assay Value, oz. per ton.
			Litharge, grms.	Sodium Carbonate, grms.	Borax, grms.	Potassium Cyanide, grms.	Charcoal, grms.	Granulated Lead, grms.					
1	Scorification.....	0.1			5.			55.	11595	0.28	1.04	1.32	11750
2	Scorification.....	0.05			5.			55.	11440	0.38	.48	0.86	11538
4	Crucible, with nail..	0.20	22.	12.5			0.7		11482	0.14	.31	0.45	11533
9	" no nail.....	0.05	15.			25.			11899	0.13	.12	0.25	11427
3	" with nail..	0.1	22.	12.5			0.7		11272	0.27	1.09	1.36	11425
5	" no nail.....	0.20	88.	11.					11375	0.10	.08	0.18	11395
8	" no nail.....	0.05	77.	5.5			0.6		11252	0.09	.12	0.21	11275
6	" no nail.....	0.20	25.			25.			11092	0.20	.19	0.39	11135

As shown in Table V., the method by scorification, using one-tenth of an assay-ton, gives the highest "uncorrected" as well as

the highest "corrected" assay, but with the exception of No. 3, carries the most silver (1.32 per cent) into the slag and cupel. Nos. 5 and 8, which contain the most litharge (77 and 85 grammes) have the least silver in the slag and cupel (0.2 per cent.); Nos. 6 and 9, with cyanide of potassium, have a little more silver (0.32 per cent) in the slag and cupel. No. 3 is a common crucible-assay, made with a nail, but only 22 grammes of litharge. The slag and cupel contain the most silver (1.36 per cent.) of all the methods.

G.—*Assaying of the Sulphides for Gold.*

In assaying for gold the sulphides should always be first treated with concentrated nitric acid and only the residue assayed. Without the preliminary treatment with acid, the amount of gold obtained by assay will usually fall considerably short of the true amount. The following is the exact method:

Weigh out into a beaker (600 to 800 c. c. capacity) 1 to 3 assay-tons of sulphides. Cover with a watch-glass and add 200 c. c. of C. P. nitric acid, at the rate of 6 or 8 c. c. at a time. Set the beaker in a moderately warm place on the sand-bath. After action has nearly ceased, add a few more centimeters of acid. If action begins again, continue to add the acid until it ceases. Dilute with about 400 c. c. of pure hot water; let stand about an hour, and filter off the undissolved residue. Dry the residue and filter-paper and burn the latter. As the residue will contain considerable silver undissolved, no more need be added on account of the gold.

H.—*Definition of Terms Used in this Paper.*

The terms "ordinary," "old" or "simple" hyposulphite solution and process, refer to the sodium or calcium hyposulphite solution and to the Kiss and Patera process in which these solutions alone are used.

The term "Russell process" includes the use of cuprous hyposulphite, acid, or any alkaline carbonate, as soda-ash, one or more of them, either without or in connection with the use of the sodium or calcium hyposulphite solution as used in the Kiss and Patera process.

"Extra-solution" properly means the cuprous hyposulphite solution of the Russell process, and, to save repetition, the two terms are used interchangeably in this paper.

The "special extra-solution" is an extra-solution which has already been used on one charge of ore and is to be used on a second

charge just previously to the use of a fresh extra-solution on that charge.

"Apparent extraction" is the extraction in percentage or in ounces by leaching in the mill, calculated from the value of the tailings and ore per ton, or, in other words, is the difference between the value of the tailings and that of the ore charged to the leaching tanks. The apparent extraction is calculated each day.

"Actual extraction" is the amount of silver actually obtained in the bullion or sulphides. When expressed in percentage, it is the amount of silver actually obtained compared with the silver in the ore actually charged into the leaching-tanks. The actual extraction is known only at the end of each leaching-run, usually of 100 to 1000 tons of ore.

Neither the apparent nor the actual extraction refers to any losses occurring previously to the charging of the ore into the leaching-tanks.

An "acid ore" as used in this paper, is a roasted ore, the first wash-water from which gives an acid reaction. Of this class are the Ontario (in 1883–84), Sombrerete, San Antonio and San Bartolo.

A "simple alkaline ore" is a roasted ore with an alkaline first wash-water, of which the Sierra Grande, Daly and Veta Grande ores are examples.

An "alkaline arsenical ore" is an ore with considerable arsenical compounds and an alkaline first wash-water. The Yedras ore is the only one of this class so far treated by the Russell process.

The "first wash-water" is used before the regular leaching-solutions and only on roasted ores, being employed to extract all salts formed in the roasting which are soluble in water.

The "second wash-water" is used after the regular leaching-solutions, on all ores, for the purpose of extracting the portion of the hyposulphite solution remaining in the ore, and also for the purpose of maintaining the volume of the stock-solution.

"Volume to saturate," for raw ores, means the volume of liquid which the charge of dry ore has absorbed per ton by the time the surface of the liquid stands level with the surface of the charge. For roasted ores, it means the volume of liquid which a charge of wet ore will absorb in the same way. By a "wet charge" is meant a charge from which water or solution has just been drained, but which still retains from 10 to 20 per cent. of moisture.

"Rate of leaching" is the number of inches depth of liquid,

measured on the surface of the charge, which will pass through that charge per hour.

"Circulation" is the re-passing of the solution (usually the extra-solution) through the charge of ore, after it has already passed through one or more times.

By "tailings" is understood any material which has been treated by leaching or amalgamation, either raw or after roasting. In this paper, the term "roasting" or "roasted ore" will include chloridizing and chloridized ore. The term "Howell furnace" will include any self-discharging revolving furnace, either partially or wholly lined with brick. All expenses are given in U. S. currency, and the value of one ounce of silver is estimated at $1.

II. Mill Work.

A.—*Preparation of the Ore for the Process.*

a. The Crushing and the Effect of Various Sizes of Crushing on the Rate of Leaching, Roasting and Mill-Results.—All crushing of raw as well as of roasted ores must be dry. Also all tailings, which are to be treated by any leaching process, must be dried if they contain any wet slimes. For ores which are to be roasted, the reason for dry crushing is apparent. In the case of raw ores, a wet-crushing causes a separation of the coarse and fine particles, which is very objectionable. Table VI gives the size of crushing for the Russell process in various mills. The extraction from Silver Reef and Raymond and Ely raw ores would probably be increased by a finer crushing.

The leaching-rate of roasted ore is greater than that of raw ore, unless the soluble salts formed in roasting have first been dissolved from the roasted ore before it is charged into the leaching-tanks, in which case the leaching-rate is about the same as for raw ores. The fineness of crushing has, within certain limits, no appreciable effect on the rate of leaching, if the ore is well chloridized and is charged dry into the leaching tanks. For instance, when treating Ontario ore, a charge of the finest flue-dust from the farthest dust-chamber of the Stetefeldt furnace leaches just as rapidly as a charge of the coarsest material from the furnace-shaft. This is due to the perfectly roasted condition of the flue-dust, which causes the charge to maintain a soft and half-floating condition during the whole of the leaching, unless it is allowed to drain and settle for a considerable time. On the

other hand, at Lake Valley, with a Howell furnace without an auxiliary fire, the flue-dust was so poorly roasted that even when mixed with the coarsest ore, it proved a great hindrance to rapid leaching.

TABLE VI.

LEACHING-RATE, SIZE OF CRUSHING, AND VALUE PER TON, OF MATERIAL TO WHICH THE RUSSELL PROCESS HAS BEEN APPLIED.

General Description of Material Treated.	Tailings or Ore.	Name and Location of Mine or Mill.	Rate Leaching. Inches per hour.	Value of Material. Oz. Silver per ton.	Size of Screen Used.	
Previously treated by other processes one to three times.	Tailings.	Leeds, etc., Silver Reef, Utah		8.	Not Re-crushed. Orig. size not known.	
	"	Bremen, Silver City, New Mexico	½	11.5		
	"	Raymond & Ely & Meadow Valley, Pioche, Nev.		8.		
	"	Veta Grande, Parral, Mexico		12.		
Raw ore.	Ore.	Silver Reef, Utah		15.	10	Mesh.
	"	Raymond & Ely, Pioche, Nevada		14.	4	"
Chloridized material with alkaline first wash-water.	Ore.	Sierra Grande, Lake Valley, New Mexico	8	12.	16	"
	"	San Miguel, Cusihuiriachic, Mexico	5	50.	12	"
	"	Yedras, Sinaloa, Mexico	7	62.	26	"
	"	Daly, Park City, Utah. (Experimental Plant)		40.	20	"
	"	Veta Grande, etc., Parral, Mexico		32.	16	"
	"	Ontario, Park City, Utah, 1887		44.	26	"
Chloridized material with acid first wash-water.	Tailings.	Sierra Grande, Lake Valley, New Mexico	8	12.	16	"
	Ore.	Ontario, Park City, Utah, 1883-4. (Experimental Plant)	10	88.	16	"
	"	Sombrerete, Zacatecas, Mexico	12	48.	8	"
	"	San Antonio, Cusihuiriachic, Mexico	10	63.	12	"
	"	Chloride, Chloride, New Mexico	5	26.	20	"
	"	San Miguel, Cusihuiriachic, Mexico	7	45.	12	"
	"	San Bartolo, " "	8	40.	12	"

NOTE.—All the above rates of leaching except for the Bremen tailings at Silver City are without syphons or vacuum.

For ores which are to be roasted in a Stetefeldt furnace, or which are to be thoroughly roasted in a Howell or reverberatory, there is practically no limit to the fineness of crushing allowable. But a roasting in a Brückner furnace may have a very bad effect on the rate of leaching, as shown further on.

For raw ores, if rolls are used, the limit of coarseness may be put at an 8-mesh screen. In the case of raw tailings, the limit of fineness allowable has only been reached in the treatment of the Bremen tailings at Silver City, New Mexico. The average fineness of that part of those tailings which could be successfully treated was such that 87.8 per cent. would pass through a screen of 150 holes to the linear inch, or 22,500 holes to the square inch. With this fineness of ma-

terial, a vacuum of 14 inches of mercury produced a leaching-rate of about one-half inch per hour, measured on the surface of the charge. These tailings came from the upper end of the Bremen tailing-pits, which, being nearest to the mill which had produced the tailings, contained the coarsest material. The tailings gradually became finer, the leaching rate decreasing until it became one-seventh to one-tenth of an inch per hour. The treatment of the tailings was then abandoned. The ore from which these tailings had been produced was originally crushed through screens of 40 to 60 meshes to the linear inch. Subsequently the tailings had been once amalgamated and once concentrated. The extreme fineness of these tailings was due to the slate and lime composing the ore, which had decomposed and weathered during the three previous treatments, and during years of exposure to air and moisture in the tailing-pits.

Table VI gives the average rate of leaching in the mills for various ores crushed through different screens and treated by the Russell process.

TABLE VII.

RESULTS OF ASSAY-OFFICE LEACHING-TESTS. ONTARIO ORE ROASTED IN A STETEFELDT FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay Office.	Per cent. Extracted from Sample taken after Five Seconds in Furnace.	Per cent. Extracted from Sample taken after Lying on Cooling Floor.
16 Screen—eight per ct. Salt added between Battery and Furnace.....	Russell Process.....	81.6	89.4
	Ordinary "	64.3	85.0
30 Screen—seven and three-tenths per cent. Salt added between Battery and Furnace.....	Russell Process.....	86.4	91.4
	Ordinary "	62.7	80.6
16 Screen—twelve per cent. Salt added between Battery and Furnace.....	Russell Process.....	90.0	96.2
	Ordinary "	76.0	90.8
20 Screen—twelve per cent. Salt added between Battery and Furnace.....	Russell Process.....	91.9	97.0
	Ordinary "	84.6	92.5
		87.5	93.5
		71.9	87.2

Tables VII, VIII, IX, X, XI, XII, XIII, XXVI, XXVII, XXVIII and XXXI illustrate the changes taking place during roasting, and the effect of various sizes of crushing and percentages of salt on the results of the furnace-work, as shown by assay-office leaching-tests. Tables VII and VIII referring to the Stetefeldt furnace, IX to the Howell, X and XI to the reverberatory, and XII, XIII and XXVIII to the Brückner, while tables XXVII

and XXXI sum up and compare the furnace-work by all the furnaces.

In regard to the size of crushing, Table VII shows that with 12 per cent. salt a 16- and a 20-mesh screen on Ontario give about the same results. These results are probably as high as could have been given by a 30-mesh. The results are given more in detail in Table VIII.

TABLE VIII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. ONTARIO ORE ROASTED IN A STETEFELDT FURNACE.

			Mill Run No. 1.	Mill Run No. 2.	Mill Run No. 3.	Mill Run No. 4.	Mill Run No. 5.	Mill Run No. 6.	Mill Run No. 7.	Mill Run No. 8.	Mill Run No. 9.
		No. of Furnace	2	2	1	2	2	2	2	2	2
		Mesh of Screen	30	30	30	30	20	16	16	16	16
		Per cent. of Salt Used	0	0	5½	9	12½	8	12	16	18
		Oz. Silver in Ton of Raw Ore	74.8	82.4	72.0	70.4	98.0	85.0	88.0	61.0	74.4
Samples Taken from Inside of Furnace Immediately After Falling. Time, Five Seconds.	"Shaft."	Oz. Silver per Ton	66.0	62.0	65.0	58.4	88.9	61.6	76.8	56.0	68.0
		Per cent. of Soluble Salts	6.5	3.5	8.5	12.0	11.5	7.8	16.0	17.2	19.0
		Per cent. by "Ord." in Assay Office	30.9	31.5	39.3	35.3	80.0	63.6	83.6	76.5	60.8
		Per cent. by "Extra" in Assay Office	66.5	50.0	82.6	89.0	89.7	77.3	90.0	84.4	86.4
	Ret. Fine.	Oz. Silver per Ton	94.0	84.0	76.8	76.0	108.0	92.4	91.2	73.0	85.0
		Per cent. of Soluble Salts	2.0	1.5	8.0	11.0	11.5	7.6	16.8	18.0	19.2
		Per cent. by "Ord." in Assay Office	29.0	15.0	44.8	31.6	88.0	65.0	68.5	86.1	78.4
		Per cent. by "Extra" in Assay Office	61.6	45.5	82.8	91.6	94.1	86.0	90.0	88.9	91.1
Samples Taken from Cooling Floor After Lying Twelve Hours.	"Shaft."	Oz. Silver per Ton	70.0	60.0	66.0	58.4	89.2	60.0	76.8	58.3	63.2
		Per cent. of Soluble Salts	8.5	6.7	12.0	12.0	12.0	8.0	16.0	17.0	20.0
		Per cent. by "Ord." in Assay Office	74.2	83.4	60.1	94.4	92.0	82.8	90.6	89.0	88.0
		Per cent. by "Extra" in Assay Office	84.0	85.8	90.0	95.2	96.9	88.5	96.0	93.5	95.2
	Ret. Fine.	Oz. Silver per Ton	.....	.....	76.8	78.0	108.0	92.8	91.1	72.3	78.4
		Per cent. of Soluble Salts	.....	.....	11.0	11.0	12.2	8.2	17.0	18.6	21.0
		Per cent. by "Ord." in Assay Office	.....	.....	76.0	92.0	93.1	83.3	91.1	94.7	93.0
		Per cent. by "Extra" in Assay Office	.....	.....	86.6	94.1	97.1	90.3	96.5	95.6	97.0

For San Antonio ore, in a Howell furnace (Table XXVII) the change from 26- to 12-meshes produces no effect; while the San Miguel results are depressed 2.8 per cent. by the same change. It should be stated that, while roasting San Antonio ore, some San Bartolo ore was always present.

Sombrerete ore (Table XXVII) in a reverberatory yields 2.7 per cent. more with an 8-mesh screen than with a 30-mesh.

Table XII apparently indicates that a 10-mesh screen gives better results than a 26-mesh on Yedras ore, with a Brückner furnace. That such is not the case, is shown by the vault samples (Table XXVIII) for the same charge, which show only 67.5 per cent. extraction or 16.6 per cent. less than the furnace sample. The "vault" referred to is the chamber into which the roasted ore falls as it comes out of the furnace. The results on another charge, also (Table XIII) show that the Brückner furnace results are 4.5 per cent. less with a 10-mesh screen than with a 26-mesh.

TABLE IX.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. CUSI ORE ROASTED IN A HOWELL FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay-Office.	Per cent. Extraction after 5 Minutes in the Furnace.	Per cent. Extraction after 15 Minutes in the Furnace.	Per cent. Extraction after 20 Minutes in the Furnace.	Per cent. Extraction after 25 Minutes in the Furnace.	Per cent. Extraction after 30 Minutes in the Furnace.	Per cent. Extraction after 35 Minutes in the Furnace.	Per cent. Extraction after 40 Minutes in the Furnace.	Per cent. Extraction after 45 Minutes in the Furnace.	Per cent. Extraction after Lying 3 Hours in Vault.
12-Screen—eight per ct. Salt	R. Process.	34.2	46.4	59.7	65.2	75.0	85.0	85.3	77.3	91.8
Two Rings—No Shelves.....	Ordinary.	7.0	9.0	12.4	.....	22.4	60.0	55.2	40.0	74.0
12-Screen—eight per ct. Salt	R. Process.	58.7	.....	59.1	62.0	72.0	60.5	60.0	61.2	92.4
Two Rings—No Shelves.....	Ordinary.	13.8	.....	18.0	17.2	50.0	22.0	28.9	29.0	76.7
12-Screen—eight per ct. Salt	R. Process.	36.5	40.0	48.6	83.0	59.0	56.0	Out of Furnace	Out of Furnace	87.3
No Rings—3 Broad Shelves...	Ordinary.	0.0	9.0	9.2	56.0	36.0	20.0			79.9
12-Screen—eight per ct. Salt	R. Process.	33.0	54.2	58.0	85.7	70.0	68.0			89.2
One Ring—3 Narrow Shelves	Ordinary.	0.0	10.0	15.0	66.7	44.2	25.0			81.7
Average,	R. Process.	39.3	46.9	56.3	73.9	69.0	67.4	72.6	69.2	90.2
	Ordinary.	5.2	9.3	12.4	46.6	38.1	31.7	42.0	34.5	78.1

Table IV shows the effect of coarse- and fine-crushing on the leaching-results in the mill. For Ontario and Sombrerete, the relation between the assay-office and mill-results remains the same whether a coarse or fine screen be used, and, for Ontario ore, whether the ore was from the shaft or the dust-chambers. But in the case of San Antonio the discrepancy between the mill and the assay-office results was increased 5.7 per cent. by a change in crushing from a 26- to a 12-mesh screen, and in the case of San Miguel ore, the same change caused an increase in the discrepancy of 3.6 per cent. As a

rule, however, a 16-mesh screen gives as good results as a 30-mesh, and the leaching-results in the mill will average within 1 per cent. of the results in the assay-office. Both the apparent exceptions given above may be due to defects in the roasting. At Lake Valley, owing to the imperfection of the screening apparatus, much ore had to be leached which would not pass a 10-mesh screen. But it was found that the material which would not pass such a screen was chloridized just as high and yielded as high a percentage in both assay-office and mill as that portion which would pass a 40-mesh. In the case of the Bremen tailings, as long as the leaching-rate remained at about one-half inch or more per hour, the difference between the mill and the assay-office extraction was only one-tenth to one-half ounce per ton.

TABLE X.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. SOMBRERETE ORE ROASTED IN A REVERBERATORY FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay-Office.	Per cent. Extraction after 4 Hours in the Furnace.	Per cent. Extraction after 5 Hours in the Furnace.	Per cent. Extraction after 6 Hours in the Furnace.	Per cent. Extraction after 7 Hours in the Furnace.	Per cent. Extraction after 8 Hours in the Furnace.	Per cent. Extraction after 9 Hours in the Furnace.	Per cent. Extraction after 10 Hours in the Furnace.	Per cent. Extraction after 11 Hours in the Furnace.	Per cent. Extraction after 12 Hours in the Furnace.	Per cent. Extraction after 13-16 Hrs & Final Per cent.
30-Screen.....	R. Process	30.7	28.0	30.2	40.8	50.0	59.1	58.4	.....	56.4	85.7
Ten Per cent Salt.....	Ordinary	7.0	14.0	7.5	7.8	11.0	18.3	27.0	.....	27.1	70.9
30-Screen.....	R Process	27.6	.....	22.4	38.0	48.2	.....	49.5	82.5	.....	87.8
Ten Per cent. Salt.....	Ordinary	11.8	.....	8.0	14.8	10.0	.....	6.0	64.3	.....	81.0
8-Screen.....	R.Process					41.2	63.4	71.1	84.3	86.7	94.3
Ten Per cent. Salt.....	Ordinary					10.0	19.9	13.8	59.6	44.2	83.7
Fifteen Per ct. Lead Ore	R.Process					38.0	.....	42.7	38.0	88.0	92.4
8-Screen, Ten Pr. ct. Salt.	Ordinary					6.0	.....	9.7	7.9	80.0	87.9
Average,	R.Process	29.1	28.0	26.3	39.4	44.3	61.2	55.4	68.3	77.0	90.0
	Ordinary	9.4	14.0	7.8	11.3	9.3	19.1	14.1	43.9	50.4	80.9

b. Effect of Various Percentages of Salt on the Roasting.—In regard to the percentage of salt for Ontario ore and a Stedtfeldt furnace, Table VII shows that either with a 16- or a 20-mesh screen 12 per cent. of salt gives the best results. The tables do not show the comparison of results caused by different percentages of salt with

TABLE XI.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A REVERBERATORY FURNACE.

Conditions Under which the Mill-Tests Were Made.	By which Process Treated in Assay Office.	Per cent. Extracted after 2-4 Hours in Furnace.	Per cent. Extracted after 5-7 Hours in Furnace.	Per cent. Extracted after 8-10 Hours in Furnace.	Per cent. Extracted after 11-13 Hours in Furnace.
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	72.5	78.2	87.0	90.0
	Ordinary "	33.9	43.4	71.6	77.9
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	63.1	72.2	86.4	87.4
	Ordinary "	25.7	37.7	66.5	78.7
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	69.2	67.9	77.5	78.5
	Ordinary "	49.9	51.8	51.0	48.6
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	.....	40.3	49.5	91.0
	Ordinary "	.....	23.4	46.6	66.3
Average,	Russell Process.....	68·3	64.6	75.1	86.7
	Ordinary "	36.5	39.1	58.9	67.9

TABLE XII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A BRÜCKNER FURNACE.

Conditions Under which the Mill-Tests were Made.	By which Process Treated in Assay Office.	Per cent. Extracted after 2-4 Hours in Furnace.	Per cent. Extracted after 5-7 Hours in Furnace.	Per cent. Extracted after 8-10 Hours in Furnace.	Per cent. Extracted after 11 Hours in Furnace.	Per cent. Extracted after 14-15 Hours in Furnace.
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	76.4	63.2	61.5	58.7	70.9
	Ordinary "	64.3	26.4	52.5	54.3	63.1
26-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	78.6	80.5	71 4	.....	.....
	Ordinary "	49.6	48.2	51.4	.....	.....
10-Screen..... } ..... Seven Per cent. Salt }	Russell Process.....	.....	79.5	80.0	84.1	.....
	Ordinary "	.....	52.9	31.1	32.5	.....
26-Screen..... } ..... Three Per cent. Salt }	Russell Process.....	60.6	82.6	86.0	85.5	85.7
	Ordinary "	35.0	49.8	48.9	33.6	13.4
Average,	Russell Process.....	71.9	76.4	74.7	76.1	78.3
	Ordinary "	49.6	44.3	46.0	40.1	38.3

a Howell furnace, but at Cusi the extraction in the assay office was not usually appreciably altered by any percentage of salt above 12 per cent. although as high as 16 per cent. was tried. The Howell furnaces used at Cusi were of cast-iron, 23 to 27 feet long and 60 inches in diameter at the lower end; the lower end, for 6 to 8 feet, being lined with brick and the remainder of the furnace being either with or without shelves, and with or without one or two brick rings to retard the ore. Tables XII and XIII show the effect of various percentages of salt on the results of the furnace samples for Yedras ore roasted in a Brückner furnace. But in this case, also, Table XXVII shows a decrease of 7 per cent. in extraction from the vault-samples, as compared with the furnace-samples.

TABLE XIII.

CHANGES TAKING PLACE DURING ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS. YEDRAS ORE ROASTED IN A BRÜCKNER FURNACE.

	No. of Charge.	Description of Charges.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. Extracted by Ordinary in Assay Office.		Per cent. Extracted by Extra in Assay Office.		Per cent. Difference Between Ord. and Extra in Assay Office.
		Results of Oxidizing Roast (Before Salt).						
Sec. I.	1	26-Screen	65.8	49.0	44.1	74.3	73.8	25.3
	2	" "	58.0	44.2		76.1		31.9
	3	" "	57.8	40.3		70.5		30.2
	4	10 "	68.8	42.8		74.3		31.5
		Results on Above Charges Immediately After Adding Salt.						
Sec. II.	1	After Adding Seven Per cent. Salt	61 2	50.1	54.9	79 2	77.9	29.1
	2	" " Five " " "	57.6	49.7		70.7		21.0
	3	" " Three " " "	.....	66.8		82.3		20.5
	4	" " Seven " " "	67.4	52.9		79.5		26.6
		Final Furnace-Results on the Same Charges.						
Sec. III.	1	Total Time in Furnace—10 Hours	61.1	52.7	40.8	74.9	69.7	22.2
	2	" " " " — 9 "	54.9	38.5		59.0		20.5
	3	" " " " — 7 "	58.9	54.3		74 4		20.1
	4	" " " " —12 "	61.0	17.9		70.4		52.5

As to the results of an oxidizing roast, the first two columns of Table VIII show the effect of such a roast in a Stetefeldt furnace on the results of assay-office leaching-tests for Ontario ore. Table XLI gives the corresponding mill-results. The first section of Table XIII shows the effect of an oxidizing roast in a Brückner

furnace on Yedras ore. The second section gives the results obtained from the same charges 15 minutes to 1 hour after the addition of salt, and section 3, the final furnace-results. The results from the corresponding vault-samples would probably be 7 per cent. less.

The comparative applicability of the various furnaces in the preparation of ore for the Russell process will be considered later.

c. *Effect on the Leaching of Wetting-Down Roasted Ore while Red-Hot.*—The effect of wetting down roasted ore on the cooling floor while red-hot may be very injurious, as shown in the following statements, the first furnished by Mr. Frank Johnson:

San Bartolo Ore, Cusi, Mexico.

	Value of Ore.	Extraction	
		by Ordinary in Assay Office.	by Extra-Solution in Assay Office.
	oz. per ton.	per cent.	per cent.
Before wetting down.....	49.1	69.1	86.3
After wetting down.....	49.1	20.3	80.3

Sierra Grande Ore, Lake Valley, New Mexico.

	Value of Ore.	Extraction	
		by Ordinary in Assay Office.	by Extra in Assay Office.
	oz. per ton.	per cent.	per cent.
Before wetting down.....	16.0	81.5	87.5
After wetting down.....	16.0	62.5	71.9

The above decrease in extraction by both the ordinary and extra-solution is probably due to the reduction of some of the silver compounds to the metallic state by the steam formed by the red-hot ore and water. No decrease occurs unless the ore is red-hot when wet down. Although the above tables give only the assay-office results, the effect on the mill-results was approximately the same. Of the above ores, the first was acid, the second alkaline, and both were roasted in Howell furnaces.

The following statement shows the effect on the Sombrerete mill-results of wetting down that ore while red-hot—the ore being acid and roasted in a reverberatory and leached in charges of 4½ tons each:

	Value of Ore.	Extraction		
		by Ordinary in Assay Office.	by Extra in Assay Office.	by Extra in Mill.
	oz. per ton.	per cent.	per cent.	per cent.
Average mill results on ore wet down while red hot.....	35.5	77.5	84.6	80.6
Average mill results on ore *not* wet down while red-hot.....	36.5	74.0	89.3	88.0

In the above statement only the difference between the extra in the mill and that in the assay office should be noted. The other differences are probably not due to the wetting-down.

That wetting-down while red-hot is not injurious on all ores is indicated by the following statement, although it refers only to assay-office results. The ore is "alkaline arsenical," and was roasted in reverberatory furnaces. The difference in value may be due to a change in the lime-compounds.

Yedras Ore, Sinaloa, Mexico.

	Value in Ounces.	Extraction	
		by Ordinary in Assay Office.	by Extra in Assay Office.
		per cent.	per cent.
Before wetting down.....	63.1	50.9	73.6
After wetting down.....	55.9	48.6	73.7

d. Weighing and Charging the Ore into the Leaching-Tanks.—For the success of the whole plant as well as of the various leaching-operations, the weight of ore charged into the leaching-tanks must be accurately known. Otherwise, the causes of losses

and discrepancies are not likely to be discovered. By weighing the ore into the leaching-tanks, it is known at once whether the losses occur during the preparation of the ore for leaching, or during the actual extraction of the silver, and treatment of the product. For instance, at Cusi, in 1887, a loss of about 14 per cent. was caused principally by dust, but, perhaps, partly by volatilization of the silver in roasting. That this loss occurred in the preparation of the ore and not in the leaching, could not have been determined, without weighing the ore between these two operations. On the other hand, if the weight of ore charged to the leaching-tanks is determined by a merely approximate method, as by weighing only 1 cubic foot of a charge, much time and labor may be expended in searching for the cause of an apparent discrepancy, three-fourths of which may not exist. Aside from the question of the extent of the inaccuracy of this method, the fact that it is at all inaccurate is demoralizing, and discourages the making of comparative experiments on all subsequent operations. It is not exaggerating the importance of this point, to say, that, upon the care used in weighing the ore charged to the leaching-tanks often depends the success or failure of the whole plant.

The manner of charging now requires attention. As the proposed method of "trough-lixiviation" does not do away with leaching-tanks, but is merely a method of charging the ore into the tanks, a treatment of that subject will not be out of place here. In this method the ore is dumped into a trough (or first into an agitator) together with water or solution. The trough conducts the material to the leaching-tanks. The result of using this method is a separating in the tank of the coarse and fine particles—the coarse particles sinking to the bottom, while a hard layer of impenetrable slimes forms on the top, if one tank is used, and the order is merely reversed if various tanks are connected. When raw ores and tailings are thus treated, no amount of vacuum (up to even 20 inches of mercury) will cause a downward leaching; and if pressure below the ore is used, the liquid simply breaks a hole through the charge at the side of the tank. If water has been used in the trough or agitator, as would be necessary in the treatment of roasted ore, it is almost impossible to introduce the solution into the ore. If solution is used first, as for raw ores, this silver-bearing solution cannot be removed. It is true that, after the supernatant solution has been allowed to settle for a long time on the surface of the charge, that

part of the solution can be decanted off, but this represents only a portion of the silver-bearing solution.

Although nearly all the chloride of silver which can be dissolved by assay-office tests with the ordinary solution, can be dissolved in the mill by the above method, so that, by taking a sample from the charge, and washing it with water in the assay office, to remove the dissolved silver, nearly all the chloride of silver can be removed, yet the method is entirely inapplicable in the mill. Another objection to the trough system is that the sulphides produced from the supernatant solutions are much reduced in value by the slimes and dirt from the ore.

The trough system was tried at Bullionville, Nev., in 1884, at Silver City, N. M., on tailings, and at Lake Valley, N. M., on roasted ore, in 1885. Although two of the above mills had been especially adapted to the method, it was abandoned for the reason above given. The sulphides produced by the trough system at Bullionville and Silver City assayed, respectively, one-third and one-eighth as much as sulphides produced afterwards by the ordinary method.

In the *Engineering and Mining Journal* of Nov. 26, 1887, the statement is made that for Cusi ore the minimum amount of ordinary or stock-solution required for trough-lixiviation was 12 parts of solution to 1 of ore, which is 384 feet per ton, although the strength of the solution was 1.6 per cent. By referring to page 52, it will be seen that the average amount of stock-solution used per ton of this ore, in the usual method of leaching, for a period of 9 months and with a solution of only 0.75 per cent., was only 80 cubic feet. Hence the trough-lixiviation, using a solution twice as strong, and requiring 4.8 times the volume, necessitated the use of 9.6 times as much hyposulphite per ton of ore. One of the evil results of this would be the production of a lower grade of sulphides and an excessive use of precipitant, as the amount of lead dissolved is directly proportional to the amount of hyposulphite used. By referring to the account of some Yedras experiments in the same journal of Jan. 21, it will be seen that the injurious effect of caustic lime was the same while using trough-lixiviation as while leaching by the usual method, even though the tailings from the former were subjected to a subsequent treatment in the assay office.

By referring to a statementmade elsewhere in this paper it will be seen that the rate of decomposition of hyposulphite solutions is directly proportional to the extent of surface exposed, so that the

decomposition of the solution will be greater for trough-lixiviation than for the usual method.

In fact, aside from the numerous disadvantages, actual experience indicates that the sole advantage, if any, obtained by this method of lixiviation is the saving of 12 to 15 cents per ton expense of moving the ore from the cooling-floor to the ore-tanks.

In brief, the objections to the method are:

1st. In the case of raw ores and tailings and of most roasted ores, the difficulty of removing the dissolved silver mechanically retained in the ore.

2d. The production from the supernatant solution of sulphides of much lower grade than those produced from material charged dry into the tanks.

3d. The difficulty experienced in first extracting the soluble salts from roasted ore (washing).

4th. The great increase in the amount of stock-solution used per ton of ore and its greater deterioration.

Returning to the general subject of charging, we have to consider the proportion between depth of charge and depth of leaching-tank. As explained further on, in the treatment of raw ores, the extra-solution is made up in a separate tank, hence no allowance is made for the making up of an extra-solution in charging the tanks. Consequently, in charging raw ores, the tanks may be filled level full, as a charge, 5 feet in depth, will certainly sink 4 or 5 inches, this space being advisable in order that any solution running upon the top of the charge shall not frequently run over. But, in charging roasted ore, sufficient space above the charge is left for the making-up of the extra-solution, unless it is necessary to run the leaching-tanks to their utmost capacity, in which case the leaching-tanks are charged level full and the extra-solution is made up in a separate tank as for raw ones. Generally, for roasted ores the depth of the extra-solution is about 34 to 46 per cent. of the depth of the ore at the end of the first washing with water. These are the proportions for roasted ores which sink about 2 to 4 per cent. during the washing.

The rate of leaching is not diminished by an increase in the depth of charge. For instance, at Cusi was one large tank, into which the roasted ore was charged to the depth of nearly 6 feet, while in all the other tanks the depth of charge was only 22 to 24 inches. Yet the rate of leaching was greater for the 6-foot charge. Also, in the treatment of the raw Bremen tailings, the extreme fine-

ness of which has already been mentioned, the rate of leaching for charges 50 inches in depth, was the same as for only 24 inches. The cause of this is that the slight increase in average compactness of the charge, due to increase in depth, is more than balanced by the increase in "head" of the liquid in the ore. Of course, it is understood that, in each case, the depth of the liquid above the surface of the charge is about the same.

That large and deep charges are the most economical, appears from the following facts: The time of leaching a 50-ton charge of ore is the same as for a 10-ton charge, if the depth of ore is the same, and is comparatively but little increased by an increase of two or three times in the depth. Less water per ton is required for a deep charge, as a given volume of water will dissolve more salts in passing through a 6-foot than a 2-foot charge. The same is true of the leaching-solutions with regard to the dissolving of the precious metals. The result of using deeper charges is a decrease in the amount of chemicals required per ton, particularly when using the extra-solution. The labor also is much decreased, the difference between 22 nine-ton tanks (as at Cusi) and 6 or 7 fifty-ton tanks would be a decrease of about 50 per cent. in the labor connected with the leaching operations. The decrease in assay-office work would be about 60 per cent. The amount of stock-solution required is also much less, which, as pointed out later on, is an important item. In fact the difference in expenses, in favor of 50-ton as compared with 10-ton charges, is not less than 65 cents per ton. For dry regions the decrease in the amount of water required is an important item.

B.—*The Chemicals.*

a. Conditions regulating the amounts used. As shown in Table XIV, the weight per ton of ore of each chemical required varies considerably. The circumstances governing the variation are as follows:

1. *Hyposulphite of Soda.*—The amount required per ton varies:

First, *according to the condition of the ore, i.e.*, whether raw or roasted. The weakening of the hyposulphite solution is chiefly due to its dilution with water while it is following the first wash-water through the ore, as in the treatment of roasted ore, and while the last wash-water is following the solution in the treatment of both raw and roasted ore. As no first, but only the last, wash-water is required in the treatment of raw ore, while both are necessary in the case of roasted ore, it is evident that the weakening of the solution

in this way is twice as great for roasted as for raw ore. Also, raw ore requires less hyposulphite for another reason. In treating roasted ore, the extra-solution, being always preceded by the first wash-water and nearly always by the stock-solution, must necessarily be used on ore containing a liquid different from itself, with which it is diluted and weakened in passing through the ore. In making it up for roasted ore an additional amount of hyposulphite must, therefore, be used. But in treating raw ore or tailings, the extra-solution is nearly always used first, and therefore on material which is perfectly dry, and no allowance has to be made for dilution.

Secondly, *directly as the volume of stock-solution.* The hyposulphite of soda is added to the solution at the rate of a certain number of pounds per ton of ore. As the principal object is to maintain the strength of the stock-solution at a certain percentage, say, $1\frac{1}{2}$ per cent, and yet use the least possible quantity of hyposulphite per ton of ore, it is evident that the smaller the volume of stock-solution the less hyposulphite will be required. For instance, if, in works treating 100 tons per day, the amount of stock-solution were 3000 cubic feet, the amount of hyposulphite required to raise the strength of that solution by any given percentage would be twice as great as that required for a stock-solution of only 1500 cubic feet. Therefore, the amount of stock-solution should be as small as possible.

Thirdly, *directly as the strength to be maintained.* Evidently, the amount of hyposulphite per ton of ore required to maintain a certain strength of solution, will depend upon what that strength is to be. For instance, each pound of hyposulphite will decompose to about the same extent, whether it is in a strong or weak solution. Therefore, in a given time, twice as many pounds will decompose in a 2 per cent. as in a 1 per cent. solution.

Fourthly, *inversely as the richness of the ore in metallic compounds dissolved by the solution.* In the dissolving of the metals of the ore by the leaching solutions, little or no hyposulphite is lost or destroyed, as the S_2O_3 of the sodium hyposulphite attaches itself to these metals only temporarily, and again returns to combination with sodium as sodium hyposulphite, when the metals are precipitated with sodium sulphide. Also, in the manufacture of sodium sulphide, hyposulphite of soda is formed as a by-product, which enters the leaching-solution when the sodium sulphide is used as a precipitant. As the amount of sodium sulphide used in precipitating a given volume of solution, and consequently the amount of hypo-

sulphite entering that volume, varies as the metal contained in it, and as the ordinary solution, in passing through a charge, seldom or never dissolves such an amount of metal as to become saturated, but approaches nearer to that point in proportion to the richness of the ore in silver, copper and (to a certain extent) lead compounds soluble in the solution, the increase in strength of that volume of solution will depend upon the richness of the ore. If only a weak solution is to be maintained, this addition to the strength of the hyposulphite solution may be sufficient to make up for the weakening from all causes.

Fifthly, *inversely as the capacity of the works.* Although the strength of the solution makes a difference in the volume of solution required to leach a ton of ore, it makes little difference in the amount of stock-solution required for works of a given capacity. It is found in practice that, with ore-tanks and precipitating-tanks of the proper dimensions, the volume of stock-solution required is less in proportion for works of 100 tons per day than for those of 50 or 25 tons. Therefore, it is evident that more hyposulphite per ton of ore will be required to keep up a given strength in works of small, than in those of large, capacity.

Sixthly, *according to the order in which the charges are treated.* The volume of stock-solution and, consequently, the amount of hyposulphite required to maintain its strength, depends upon whether the charges are treated in rotation and with regularity, or so irregularly that the number of charges requiring solution varies considerably at different times. Regularity in the treatment of charges is of importance for other reasons also.

Seventhly, *with roasted ore, as the ore is acid or alkaline.* The difference in weight of chemicals required by acid and alkaline ores is very marked. Below is given the amount of hypo used in each case per ton of ore:

Acid Ores.		Simple Alkaline and Arsenical Alkaline Ores.	
Ontario	5 lbs. Hypo.	Veta Grande	4.0 lbs. Hypo.
San Antonio	7 " "	Sierra Grande	2.6 " "
Sombrerete	5.3 " "	San Miguel	3.7 " "
San Bartolo	3.0 " "	Daly	1.5 " "
Average	5.06 lbs.	Average	2.9 lbs.

The difference is due principally to the fact that both extra and ordinary solutions may be used much weaker for simple alkaline than for acid-roasted ores.

2. *Blue-Stone.*—The amount of blue-stone used per ton of ore depends

First, *upon the condition of the ore, i.e., raw or roasted.* As shown in Table XIV., the average amount of blue-stone for raw ores is 2.63 lbs., and for roasted ores 5.6 lbs., per ton.

Secondly, in the case of roasted ores, upon *the acid or alkaline character of the ores.* For the four acid ores, Ontario, San Antonio, Sombrerete and San Bartolo, the amount of blue-stone per ton is 6.8 lbs. On the other hand, for the four alkaline ores, Lake Valley, San Miguel, Veta Grande and Daly, the average is only 4.4 lbs. per ton of ore.

Thirdly, *upon the method of using the extra-solution.* If the extra is preceded by the special extra-solution, the amount of blue-stone required per ton is considerably diminished.

3. *Caustic Soda and Sulphur.*—The weight of sulphur used in making sodium sulphide is always two-thirds of the weight of caustic soda. The weight of caustic soda and, therefore, of sulphur, depends:

First, *upon the richness of the ore in metallic compounds soluble in solution, but not soluble in water.* It is evident that the amount of precipitant required per ton will, other things being equal, vary as the amount of silver to be precipitated or directly as the value of the ore in silver. Of the forms of copper occurring naturally, only one, so far as known, namely, carbonate of copper, is soluble in solution, and at the same time insoluble in water. As it is not likely to occur in roasted ore, it is only of importance in this connection when it occurs in an ore treated raw. Dichloride of copper occurs to a small extent in chloridized ores and is soluble in the hyposulphite solution, but practically insoluble in water. Sulphate of lead occurs in roasted ore and sometimes in raw ores, and is slightly soluble in hyposulphite solution. The amount, in excess of 0.5 per cent., present in the ore is immaterial, as only a few pounds in any event would be dissolved during the leaching.

Secondly, *upon the amount of blue-stone used, and the mode of using it.* Evidently, any unnecessary increase in the amount of blue-stone used per ton should be avoided; for, with the exception of that part of the copper taking the place of the silver dissolved by the cuprous hyposulphite, and the small part expended in counter-

acting the effect of caustic soda or lime, the copper is precipitated by the sodium sulphide, increasing the consumption of the precipitant, and lowering the grade of the sulphides. As the amount of blue-stone used per ton of ore varies as the ore is raw or roasted, and (with roasted ore) as it is acid or alkaline, the amount of caustic soda is, indirectly, dependent upon the same conditions, the amount used being less for raw than for roasted ores, and less for alkaline than for acid ores.

4. *Soda Ash.*—The amount of soda ash used per ton of ore depends

First, *upon the presence or absence of sulphate of lead.* The amount of sulphate of lead in an ore (when over 0.5 per cent.) makes, for reasons already mentioned, but little difference in the consumption of soda ash.

Secondly, *upon the amount of sulphate of lime in stock-solution.* If the stock-solution is saturated with sulphate of lime, its dissolving power for sulphate of lead is decreased, approximately, one-half. The presence of such a percentage of sulphate of lime, would, however, prevent the economical use of soda ash for the precipitating of the remaining half, and is undesirable in other ways. In treating lime ores, soda ash is not generally used.

Thirdly, *upon the temperature of the solution.* The amount of soda ash required, varies as the temperature of the solution, since lead sulphate is more soluble in hot than in cold solutions.

5. *Sulphuric Acid.*—Acid is used for two purposes, the acidifying of the first wash-water for roasted ores, and the neutralization of caustic impurities in the solutions. Its use is confined almost entirely to the treatment of roasted ores. The weight of acid used per ton depends:

First, *upon the state of the ore, i.e., raw or roasted.* As raw ores have no first wash-water to be precipitated, and no caustic alkali is present in raw ores, it can enter the solution only by means of the sodium sulphide. This amount is so small that it is generally neutralized by the extra-solution, so that, when extra-solution is used, no acid is required in the treatment of raw ore.

Secondly, *upon the acidity or alkalinity of the first wash-water.* In treating roasted ores, the silver is usually precipitated from the first wash-water by means of acid and old iron. The amount of acid required per ton varies according to the condition of the first wash-water. If it be alkaline, more acid must be used than if it were neutral or acid. The state of the first wash-water varies with the

nature of the ore. Very base ores often produce such an acid wash-water, that no further addition of acid is necessary. But "free" ores usually produce a neutral or alkaline wash-water. Often, even when the wash-water is quite alkaline, the amount of silver dissolved from the ore by it will be more than one-fourth of the silver in the ore, and sometimes (as at the Sierra Grande) mere dilution with water will fail to precipitate it, nor will precipitation take place on organic substances within a reasonable time, nor on iron or copper until the addition of sufficient acid.

Thirdly, *upon the amount of silver extracted by the first wash-water*. Often, as at the Ontario with a Stetefeldt furnace in 1883, and at Sombrerete with a reverberatory furnace, the amount of silver dissolved by the first wash-water is so small that it is not precipitated, so that no acid is required.

In other cases, as at the Sierra Grande, the amount may be 30 per cent. of the value of the ore, and 2 pounds of acid may be required.

Fourthly, *upon the presence or absence of caustic lime in the ore.* The presence of caustic lime in the roasted ore usually doubles the amount of acid required for precipitating the wash-water, making it 2 pounds for such ores, as against 1 pound for ores producing a neutral wash-water.

Fifthly, *upon the mode of using the extra-solution.* Hydrate of copper is practically insoluble in the ordinary solution, so that the extra-solution acts as well as an acid as far as the neutralization of caustic alkali is concerned. But as the blue-stone is usually added to only a small portion of the ordinary solution, its effects are confined to that portion. So, in the case of alkaline roasted ore, where the amount of caustic alkali is always greater than for raw ores, it might seem that the blue-stone used would not be sufficient to neutralize all the caustic alkali, and that the use of acid might become necessary to the extent of ½ to 1 pound per ton. This, of course, would be the case in the treatment of roasted ore containing caustic lime, if it were not for the fact that the extra-solution is usually employed on such ores in such a way as to counteract entirely the effect of the caustic alkali. If the ordinary solution is used first, before the extra, the caustic alkali acts injuriously upon the silver compounds dissolved by it. For this reason, the extra-solution is used first, and in such a volume that, although weak, it protects the silver from the action of the caustic alkali.

Sixthly, *upon the mode of precipitating the first wash-water.* Of

course, the amount of acid used will depend upon the mode of precipitating the first wash-water, whether by that method just described, or by dilution, or by sodium sulphide.

b. Description and Cost:

1. *Sodium Hyposulphite* is packed in barrels of two sizes, one containing about 112 pounds, and the other 550 to 600 pounds. It does not decompose in contact with the atmosphere. If imported in large lots from Europe, it costs in New York 1.8 cents per pound. If bought from dealers, the price asked is 2¼ cents per pound in New York, 2.5 cents in St. Louis, and 3 cents in San Francisco.

2. *Caustic Soda.*—Only a high grade should be ordered, of 70 to 77 per cent., on account of freight-charges. It is imported principally from England put up in sheet-iron drums, holding about 630 to 700 pounds, usually 640. It should not be left exposed to the air more than a day or two after being broken up, especially in hot weather, as it gradually decomposes on the surface, owing to absorption of carbonic acid. The price in new York is 2.425 cents per pound for 60 per cent. Hence, a 74 per cent. caustic soda would cost three cents per pound.

3. *Sodium Carbonate.*—Although any carbonate of soda can be used, the bicarbonate is less suitable than soda ash, on account of cost, and for other reasons. It is most economical to buy the form of soda ash known as "pure alkali" of 58 per cent. made by the Solvay or ammonia process. It is made in this country at Syracuse, New York, and costs 1.4 cents per pound at the works. This alkali, in the form of a white powder, is so pure that it contains 98.7 per cent. of sodium carbonate, the remainder being principally sodium chloride and sulphate, and 0.25 per cent. moisture. On account of the entire absence of the sodium sulphide and caustic soda (which are always formed in the ordinary soda ash), the solution for the precipitation of lead does not require purification. Both caustic soda and soda ash may be purchased of Wing and Evans, N. Y. city.

4. *Copper Sulphate or Blue-Stone.*—Price per pound 4¼ cents in New York.

5. *Sulphur.*—It is immaterial whether the sulphur is crude or refined, lump or flower of sulphur. The lump-sulphur requires to be broken up only so that it will pass a one-inch mesh screen. The native sulphur, occurring in large quantities in Utah, Nevada and other places, is suitable. The price in New York of refined sulphur

in rolls is 2⅜ cents per pound; flower, 2⅝ cents; Virginia rock, 2½ cents.

6. *Sulphuric Acid* of 66° B. is best transported in iron tanks, which hold about 1700 pounds. If shipped in this way, the freight is much less than in glass carboys. The cost of a sheet-iron tank is $6.50. Price in New York per pound 1.25 cents; in St. Louis and San Francisco, 1.75 cents. Shipped in car-loads the freight, for instance, from San Francisco to New Mexico, is only 1.67 cents per pound.

c. *Comparison Between Sodium Sulphide and Calcium Sulphide.*— If the lead is precipitated by itself by means of soda ash, sodium sulphide, and not calcium sulphide, should be used as a precipitant for the silver. But in any case sodium sulphide is far better as a precipitant than the calcium salt, for the following reasons:

First, the precipitating power of a pound of sulphur combined with sodium is nearly three times as great as when combined with calcium. For in the preparation of sodium sulphide the monoq-sulphide (Na_2S) is formed, but in making calcium sulphide, sufficient sulphur must be used to form the pentasulphide CaS_5, as the lower sulphides are practically insoluble in water and therefore cannot be used.

Secondly, the solubility of calcium hydrate as stated in works on chemistry, is such that at 100° C., 1300 parts of water dissolves only one part of the hydrate, while the same amount of water at the same temperature dissolves 2600 parts of caustic soda. The solubility of caustic soda being thus over 3000 times as great as that of lime, it is not strange that the time required in making calcium sulphide is many times that required for sodium sulphide.

Thirdly, the amount of heat required in the preparation of sodium sulphide is only 2 per cent. or 3 per cent. of that for calcium sulphide. This is due not only to the fact that the union between sodium and sulphur takes place almost instantly, but also because there is a great rise in temperature due to this chemical action which causes the temperature of the mass to rise far above the boiling point of water.

Fourthly, the sodium sulphide is over twenty times as strong as the strongest calcium sulphide solution which can be made, that is, a given volume of the former will precipitate twenty times as much as the same volume of the latter. Hence there is a less dilution of the stock solution during precipitation.

Fifthly, as shown not only by the formula but in practice, the

product obtained while using calcium sulphide is contaminated with over three times as much free sulphur as that obtained while using sodium sulphide. The amount of sulphur in the sulphides at Cusi and Sombrerete (using sodium sulphide) is so small that the difference in weight and value between dried and roasted sulphides is only 6 per cent., thus allowing the substitution of a mere steam drying for the roasting, during which latter operation some silver is always lost by dust and volatilization.

Sixthly, if calcium sulphide is used, the product is contaminated with calcium sulphate also (the sulphuric acid being derived from sodium sulphate always present in the lixiviation solution), which decreases the value of both dried and roasted sulphides and increases the weight and expense for shipment and refining.

Seventhly, the extent and cost of plant required for making calcium sulphide is much greater. For instance, at Cusi while making calcium sulphide, four iron tanks were in constant use, three of which were always boiling.

For the preparation of sodium sulphide only one of these tanks was used, and for three hours only (using steam 1 hour), once in two and one-half to three days. The above comparison illustrates also the difference in time, labor and heat required.

Eighthly, the continual use of calcium sulphide gradually converts a sodium hyposulphite solution into one of calcium hyposulphite, which has been claimed as a better solvent for gold than the former, although no one has made the statement from his own experience, or has produced any experiments or statistics. On the other hand it has been repeatedly demonstrated on the small scale, and at Cusi on the large scale, that the solvent energy of the two solutions for gold is just the same.

Ninthly, the calcium hyposulphite solution decomposes much quicker than sodium hyposulphite, and, as shown in actual practice, a greater addition of hyposulphite to the solution is required to maintain it at 1 per cent. when calcium than when sodium hyposulphite solution is used. As shown in many laboratory experiments at the Ontario, the average rate of decomposition of calcium hyposulphite solution was from 5 to 13 times as great as for sodium hyposulphite. In these tests, solutions of 2, 5 and 10 per cent. were exposed in soup plates at a temperature of 65 to 70 Fah. for seven days.

Tenthly, not only is the use of calcium hyposulphite disadvantageous as just described, if a strong solution is to be maintained, and,

as shown further back, more expensive in time, labor and fuel, but also the actual cost of chemicals is greater per ton of ore treated. For instance, at Cusi, the comparison between a run of 1900 tons of ore, using sodium sulphide, and 1000 tons, using calcium sulphide, showed that the expense of chemicals alone required for calcium sulphide was 34 per cent. greater per ton of ore treated than for sodium sulphide. If the cost of the labor and heat required were included, the difference would be still greater. At Sombrerete the comparison was made by Mr. Watson under the most unfavorable conditions for sodium sulphide, the cost of a pound of caustic soda at that place being over 30 times as great as caustic lime, yet the result of the test showed the superiority of the sodium sulphide.

d. The Objects of the Use of Sodium Carbonate or Soda Ash, as a Precipitant for Lead are:—First, the obtaining of a precious metal product entirely free from lead, thus decreasing the weight of product and cost of transportation, and, on the other hand, increasing the grade, and, to some extent, the rate paid per ounce of silver. At the Ontario, using a 1½ per cent. stock-solution, the grade of the product was increased over 100 per cent. by the use of soda ash, and the weight decreased 52 per cent., thus effecting a great decrease in express charges. As the silver in sulphides is generally paid for at the rate of 97 to 99 per cent. less $100, more or less, for treatment, and, as this $100 is 1 per cent. of $10,000, but 2 per cent. of $5,000 sulphides, there is a saving of 1 per cent. if the sulphides assay the former instead of the latter.

Secondly, the obtaining of the lead product pure from other metals, and in a more compact and marketable shape than as sulphide. The separation of lead by means of soda ash is so complete that not the slightest trace is left in the solution after precipitating, and the solubility of carbonate of silver, gold and copper is such that no trace of these metals is found in the lead precipitate, except silver to the extent of one two hundredth part of 1 per cent.; at the same time no other metals can be precipitated with the lead carbonate, as all the compounds of zinc, manganese, nickel, cobalt, iron, etc. which are in the least soluble in a hyposulphite solution are perfectly soluble in the first wash-water and are removed by it. The precipitate of lead carbonate is heavy and compact, occupying only about one-quarter of the bulk of the same amount of lead in the form of a sulphide. The precipitate is pure white, and can be sold as it is for about the cost of precipitation or converted into litharge by exposure to a low heat.

Thirdly, the securing of a great saving in the amount and cost of caustic soda and sulphur used per ton of ore, and the ability to use solutions of any temperature and strength, which would otherwise be impracticable on account of the greatly increased amount of sodium sulphide which would be required to precipitate the greater amount of lead dissolved. The solubility of sulphate of lead in the hypo-sulphite solution increases directly as the strength and temperature. For this reason, the leaching by the old process is usually done with weak cold solutions, but, if the lead is precipitated by soda ash, the amount of lead dissolved is immaterial, and the solution can be used at any strength and temperature. Besides, the cost of precipitating a pound of lead by soda ash is much less than by means of sodium sulphide. At the Ontario, using a stock solution of 1½ per cent. the net saving due to the use of soda ash in chemicals alone, without taking into account the value of the carbonate of lead, was 28 cents per ton of ore treated. With stronger solutions, the saving by the use of soda ash is much greater, with a 10 per cent. stock-solution being about $2.80 per ton of ore. In short, the use of soda ash in the case of most ores secures a three-fold saving, viz:—in express charges, in rate paid for silver, and in cost of chemicals required for precipitating.

C.—*General Order of Applying the Wash-Waters and Solutions.*

The order of applying the solutions is treated in detail further on. In general, the order for wash-waters and solutions is as follows:

a. For Raw Ore.—No first wash-water. A strong extra-solution is used first, either with or without circulation, followed by the ordinary, and finally by wash-water. In rare instances, the ordinary solution is used first, followed by a strong extra, circulated, then by ordinary, and lastly by wash-water.

b. For Roasted Ore with Acid First Wash-Water.—First wash-water; ordinary solution; strong extra-solution, circulated; ordinary solution; second wash-water. No exception to this rule has so far been encountered.

c. For Roasted Ore with Alkaline First Wash-Water (Simple Alkaline Ore).—First wash-water; much weak extra-solution, not circulated; ordinary solution; second wash-water. This has been the invariable order so far.

d. For Alkaline Roasted Ore with Arsenical Compounds (*Alkaline Arsenical Ore*).—First wash-water; ordinary solution; strong extra-solution, allowed to stand; ordinary solution; second wash-water.

D.—*The Wash-Waters.*

a. The First Wash-Water.—The chief difference between the lixiviation of raw and roasted ore is in the use of the first wash-water on roasted ore to remove the soluble salts formed in roasting, which would contaminate both solutions and products, if not removed before the leaching solutions are applied.

1. *Manner of Using the First Wash-Water.*—After the surface of the charge of ore has been levelled, water is turned into the tank, either upon the top of the charge or beneath it under the filter.

The first of these methods is used if little silver is extracted by the first wash-water, or if, as at Yedras, the extraction of a considerable amount of silver in this way is not objectionable. The outlet of the leaching-tank remains closed until the tank is about full of water. The outlet is then opened, and the leaching with water continued until nearly all the soluble salts are removed, as indicated by testing the liquid with sodium sulphide. The water running upon the ore is then shut off, and what remains in the tank allowed to run out. When the water has drained out of the ore, the leaching-solution is turned upon the top of the charge.

If the second method is used, which is the case if the amount of silver extracted by the first wash-water is to be diminished as much as possible, or if water is scarce and must be used economically, the wash-water is introduced under the filter and allowed to rise slowly through the charge of ore until the tank is filled. The water is then turned off, and the outlet of the leaching-tank opened. When the surface of the water in the tank reaches the surface of the charge, the water is turned upon the surface, which is kept covered until the washing is finished. The rest of the manipulation of the first wash-water is the same as that already described.

The introduction of the leaching-solution upon a charge from which the wash-water has been drained has two advantages over turning the leaching-solution upon a charge at the time the descending wash-water disappears under the surface of the ore. The first advantage is that, in practice, the stock-solution loses less in strength by this method. The second is, that the watching for the appearance of the first silver-bearing solution at the outlet of the leach-

ing-tank is avoided, as well as the danger of losing silver by carelessness at this point. For, in the method just advocated, the changing of the outlet-hose of the leaching tank from the wash-water trough or launder, to that leading to the precipitating tanks, is made at any time after the charge has drained, the leaching-solution not being used on the charge until such change has been made. For the method of using the first wash-water at Yedras, see page 490.

TABLE XIV.

AVERAGE AMOUNT OF CHEMICALS AND VOLUME OF WATER USED PER TON ON RAW AND ROASTED ORES AND TAILINGS TO WHICH THE RUSSELL PROCESS HAS BEEN APPLIED.

General Description of Material Treated.	Tailings or Ore.	Name and Location of Mine or Mill.	Pounds of Blue-Stone Per Ton.	Pounds of Hypo Per Ton.	Pounds of Caustic Soda Per Ton.	Pounds of Sulphur Per Ton.	Pounds of Lime Per Ton.	Pounds of Acid Per Ton.	Cu. Ft. of First Wash-Water Per Ton.	Cu. Ft. of Second Wash-Water Per Ton.	Total Cu. Ft. of Water Per Ton.	Total Weight of Chemicals Per Ton.
Previously Treated by Other Processes 1 to 3 Times.	Tailings.	Leeds, etc., Silver Reef, Utah..	1.8	3.76	3.00	3.00	.....	0.25	.....	10.5	10.5	11.8
	"	Bremen, Silver City, N. Mex...	4.1	3.00	1.75	1.1	.....	.25	.....	9.0	9.0	10.2
	"	R. & E. & M. V., Ploche, Nev..	2.0	1.50	1.50	1.5	.....	0.00	.....	9.5	9.5	6.5
	"	Veta Grande, Parral, Mexico.	4.0	2.75	.....	4.0	5.0	0.25	.....	11.5	11.5	16.0
Raw Ore.	Ore.	——— Silver Reef, Utah	1.8	3.76	3.00	3.0	.....	0.25	.....	10.5	10.5	11.8
	"	Raymond & Ely, Ploche, Nev.	2.0	1.50	1.50	1.5	.....	0.25	.....	9.5	9.5	6.8
Roasted Material with Alkaline First Wash-Water.	Ore.	Sierra Grande Lake Valley, New Mexico.....	2.6	2.6	1.4	0.9	.....	1.80	39.5	2.5	42.0	9.3
	"	Daly, Park City, Utah.....	4.5	1.5	.....	.....	.....	.....	.....	.....	45.0	.....
	"	San Miguel, Chihuahua, Mex.	4.7	3.7	4.0	2.7	.....	0.00	37.5	2.5	40.0	15.1
	"	Yedras, Sinaloa, Mexico.....	10.0	0.5	6.0	4.0	.....	0.00	32.0	14.0	46.0	20.5
	"	Veta Grande, etc., Parral, Mexico.....	6.0	4.0	4.5	3.0	.....	0.00	41.5	2.5	44.0	17.5
	"	Ontario, 1887, Park City, Utah	6.4	.....	.....	.....	.....	.....	.....	.....	122.0	.....
	Tailings.	Sierra Grande, Lake Valley, New Mexico.....	2.6	2.6	1.4	0.9	.....	1.80	31.5	2.5	34.0	9.3
Roasted Material with Acid First Wash-Water.	Ore.	Ontario, Park City, Utah, 83-84	7.0	5.0	5.1	3.4	.....	1.00	138.5	2.5	141.0	21.5
	"	Sombrerete, Zacatecas, Mex...	7.0	5.3	4.9	3.3	.....	0.00	47.0	2.5	49.5	20.5
	"	San Antonio, Chihuahua, Mexico.....	8.3	7.0	7.75	5.0	.....	0.00	52.5	2.5	55.0	28.0
	"	Chloride, Chloride, N. Mex....	4.3	5.5	5.25	3.5	.....	3.80	38.5	2.5	41.0	22.1
	"	San Miguel, Chihuahua, Mex.	5.0	3.7	5.0	3.4	.....	0.00	37.5	2.5	40.0	17.1
	"	San Bartolo, Mex.....	5.0	3.0	5.0	3.3	.....	0.00	37.5	2.5	40.0	16.3

2. *Volume of First Wash-Water for Various Ores.*—When the first wash-water is used as just described, the volume of the second wash-water for roasted ore is much diminished and amounts to only 1¼ to 2 cubic feet per ton. Table XIV gives the amount of water required for various ores, including the first and second wash-water and the total amount. Except in the case of the Ontario ore, the amount of the first wash-water varies from 12 to 53 cubic feet per ton of ore, the average being about 40. At the Sierra Grande, Sombrerete, and Chloride mills, the water is introduced from above. The saving in the amount of the first wash-water by introducing it below the charge is 25 to 40 per cent.

3. *Advantage of Charging Roasted Ore Cold into the Leaching-Tanks.*—As shown in Table XV, the amount of silver dissolved from roasted ore by the first wash-water varies with the temperature of the ore at the time the water is turned upon it. The term "hot dry ore" refers to ore which has a temperature of about 150° to 250° F.; "cold dry ore" to ore at a temperature of not more than 120° F. Of the silver dissolved and extracted from the ore by the first wash-water, at Cusi, only a portion actually left the leaching-tank. As shown in Table XVI, about 90 per cent. of all the silver extracted by the first wash-water is extracted during the first fifteen minutes' leaching, provided the water is introduced on top of the charge, as was the case in Tables XV and XVI.

As more or less water, comparatively free from salts, may exist under the filter at the time the first wash-water is used, being left there by the last wash-water of a previous charge, and as chloride of silver is precipitated from an aqueous solution by dilution with water, a large portion of the dissolved silver may be precipitated from the first wash-water, under the filter, and therefore not actually leave the leaching-tank until again dissolved by the hyposulphite solution subsequently used. For Cusi ore, even when 25 to 30 per cent. of the silver in the ore had been dissolved and extracted from the ore by the first wash-water, only 3 to 9 per cent. actually left the leaching-tank, while 2 to 6 per cent. left the tank when the ore was charged cold. Consequently, the difference between the results of hot and cold charging are not practically as great as would appear from Table XV. As far as the fineness of the wash-water precipitate and (except in some parts of Mexico) the cost of the precipitating is concerned, the amount of silver passing out of the leaching-tank in the first wash-water is of little importance. For instance, at Lake Valley all the silver dissolved by the first

wash-water, amounting to 30 per cent. of the value of the ore, actually passed out of the leaching-tanks in the first wash-water; but

TABLE XV.

SILVER DISSOLVED FROM COLD AND HOT ROASTED ORES BY THE FIRST WASH-WATER.

No. of Charges Experimented on.	Value of Ore in oz. Silver Per Ton.	Per cent. of Salts Soluble in Water.	Calculated Value in oz. Silver Per Ton.	Value after Washing in Mill. Oz. Silver Per Ton.	Ounces Extracted by First Wash-Water.	Per cent. Extracted by First Wash-Water.
HOT DRY ORE.						
5	34.6	14.8	40.6	28.3	12.3	30.3
33	34.2	14.1	38.8	27.6	11.2	28.8
COLD DRY ORE.						
19.	37.9	15.0	44.6	39.4	5.2	11.6

all of it was saved except 0.07 ounce per ton of ore. The only objection to allowing a large amount of the silver in the ore to pass out of the leaching-tank in the first wash-water is, that this is the only liquid, used in leaching roasted ores, which is thrown away or allowed to pass out of the mill; and, consequently, incomplete precipitation would cause a loss which could not afterwards be remedied; while no loss would result from incomplete precipitation of the regular leaching solutions, as any silver not precipitated at one time would be precipitated subsequently—the same solution remaining in constant use.

4. *Amount of Silver Extracted by the First Wash-Water from Various Roasted Ores.*—Table XVII gives the amount of silver extracted from the first wash-water. The actual loss by unprecipitated silver in the first wash-water, was only 0.07 ounce per ton for Sierra Grande ore, although 3 to 4 ounces were dissolved by it and passed out of the leaching-tank. The reason why more silver is dissolved by the first wash-water from some ores than from others is not known. There seems to be no connection between the amount dissolved and the percentage of salt, or the kind of furnace used. Even while using 17 per cent. of salt, at the Ontario, on ore roasted in a Stetefeldt furnace, only 0.25 per cent. of the silver was extracted by the first wash-water, while 30 per cent. was extracted from the Sierra

Howell furnace, and
st wash-water from
t in a reverberatory.

RST WASH-WATER

Silver Contained in 1000 c.c. at end of Four Hours.	Silver Contained in 1000 c.c. at end of Five Hours.	Silver Per 1000 c.c. at end of First Thirty Inches of Water.	Total Time of Washing.
.0008	None.		19 Hours.
Trace	"		15 "
.0008	.0004		16 "
Trace	None.		18 "
None.	"		15 "
		.0038	

mmes per 1000 c.c.

the First Wash-Water.

rammes per 1000 c.c.
ng applied on top of
of the latter method
d, but causes it to be
ter.) The table indi-
d silver passes out of
ng.

ater on Ontario ore in
was introduced first
l, after which all the
pitated on top of the
ter coming from the
t was precipitated by
ved by the first wash-

water was found in the second part. If the leaching had been downward from the beginning, probably 75 to 88 per cent. of the dissolved silver would have been found in the first third of the wash-water.

TABLE XVII.

PROPORTION OF SILVER EXTRACTED FROM ROASTED ORES BY THE FIRST WASH-WATER: ITS VALUE, AND THE COST OF PRECIPITATING IT.

Name of Mine.	Wash-Water Introduced from Above or Below.	Mode of Precipitating.	Per cent. of Ore-Value Obtained from Wash-Water.	Value of the Precipitate in Oz. Per Ton.	Cost of Precipitating Per Ton of Ore.
1883.					
Ontario	From Below.	Not Precipitated.	0.25	Not Saved.	Not Precipitated.
Lake Valley	From Above.	Acid and Old Iron	29.	9000 oz.	9 cts.
Sombrerete	" "	Not Precipitated.	Trace.	Not Saved.	Not Precipitated.
San Bartolo	From Below.	Sodium Sulphide.	8.3	1600 oz.	12 cts.
San Antonio	" "	" "	6.4	1400 oz.	15 cts.
San Miguel	" "	" "	7.8	2400 oz.	15 cts.
Yedras	" "	" "	2.0	13934 oz.	6 cts.
Chloride		Acid and Old Iron			16 cts.

The wash-water being acid, 72 per cent. of the dissolved silver was obtained with the cement copper, by simply hanging old iron in the wash-water precipitating-tanks. The addition of ½ to 1 pound of acid to the wash-water might have been necessary, if the wash-water had contained much silver. But in this case it was not necessary, as the amount of silver dissolved by the first wash-water was only a fraction of an ounce per ton of ore, and all of it was regained except about one-twentieth of an ounce per ton.

The difference between the Ontario and Lake Valley ores, as to the percentage of silver dissolved by the the first wash-water, has already been remarked. If the actual ounces per ton of ore extracted in this way are compared, the difference is not quite as great, but is still about 6 or 7 times as great for Lake Valley, as for Ontario.

The precipitate from the first wash-water (Ontario) by means of old iron, contained about 73 per cent. copper, the amount of copper obtained per ton of ore being about 3¼ lbs., although the raw ore contained only about ⅛ of 1 per cent. or 7 lbs. per ton. The amount of silver and copper in the first wash-water at the Ontario in 1883–1884, was as follows:

Copper in the precipitate by iron, 73 per cent.; weight of copper obtained per ton of ore, 3.25 lbs.; grammes silver contained in 1st 40 cubic feet of wash-water, 6.0; grammes silver contained in 2d 40 cubic feet of wash-water, 31.0; grammes silver contained in 3d

40 cubic feet of wash-water, 2.6; proportion of ore-value leached out in 1st wash-water, 0.85 per cent.; proportion of wash-water silver obtained with the copper precipitated on iron, 72 per cent.

6. *Precipitation of the Silver from the First Wash-Water.*—Besides the method of precipitating by means of acid and old iron, the silver may also be precipitated from the first wash-water by means of sodium sulphide, or sometimes by dilution. The method by sodium sulphide is probably the better for Mexico of the first two, and is quicker, as the precipitation of the silver by iron requires about 12 hours, and heating of the wash-water in the precipitating-tanks. The average value of the various precipitates from different ores by the two methods would probably not differ much. But the method by acid and iron seems to be the most thorough, except when the wash-water contains such a very small amount of base metals, that (as at Yedras) all the base metals can be economically precipitated along with the silver. This very rarely happens. In most cases, when sodium sulphide is used, only enough is added to precipitate all the silver and part of the copper, without precipitating much of the remaining metals; but sometimes a small amount of silver remains unprecipitated. A point in favor of the method by acid and iron is, the extremely small bulk of the precipitate compared with that produced by sodium sulphide. This method consists in suspending old iron in the wash-water tanks and heating the water to about 175° F. (more or less), after acidifying it with 1 to 2 pounds of acid per ton of ore. The method of "dilution" consists in simply diluting the wash-water with sufficient pure water to precipitate the chloride of silver. It is not a safe method, however, for some ores, and it requires considerable water.

7. *Value of the Precipitates from the First Wash-Water.*—Table XVII gives the value of the precipitates, by both methods, from the wash-water of various roasted ores. The value of the precipitate of either method depends mostly upon the amount of copper present in the first wash-water, as it is practically impossible to avoid precipitating copper by either method. As a rule, the precipitate from ores with an alkaline wash-water is much the higher in value per ton.

b. The Second Wash-Water and Mode of Using it.—The volume of the second wash-water per ton for raw ores is always the same as the volume required to saturate the wet charges. It is used to restore the volume of the stock-solution, which would otherwise be diminished, for each charge treated, by an amount equal to the stock-

solution remaining in the ore when the charge is drained. This amount is about 5½ cubic feet per ton of ore.

For raw ores there are two methods of applying the second wash-water. In the first the ordinary solution is left standing level with the surface of the charge, the outlet of the leaching-tank being stopped. A measured number of inches of water, equal to the amount required to saturate the wet charge, is then run upon the charge. The outlet of the leaching-tank is then opened and the leaching continues until the water is at the surface of the charge. The water remaining in the charge is then allowed to run to waste, but not into the precipitating tanks for the leaching-solutions. By the second method employed, if water is scarce and must be economically used, the leaching-solution is allowed to drain out of the charge, and is then followed by water to the extent of about 5½ cubic feet per ton of ore, all of which is allowed to pass into the stock-solution. The volumes of second wash-water given in Table XIV for raw ores and tailings are those required when the first method is used. The average amount required for the six raw ores and tailings by the first method is 10.0 cubic feet per ton of ore. The use of the second method would reduce this to about 5½ cubic feet per ton.

The mode of using the second wash-water for roasted ores is the same as for raw ores, if the descending first wash-water was followed by solution at the time the surface of the first wash-water reached the surface of the ore. But if the first wash-water has been drained from the charge before the solution is used, then likewise the solution must be drained from the charge before the second wash-water is used. In this case the second wash-water is then applied by suddenly flooding the surface of the charge with about one-half inch of water, which descends through the charge and is allowed to pass into the stock-solution.

c. The Total Volume of Water Required for both Raw and Roasted Ores.—Table XIV gives the total volume of water required per ton of ore for both raw and roasted ores. For all the roasted ores, including Ontario, the average is 50 cubic feet. The average volume for all the raw ores and tailings is 10½ cubic feet, if the first, and 5½ if the second method is used.

E.—*The Leaching-Solutions.*

a. Volume, Strength, Temperature, Preparation, and Description of the Ordinary or Stock-Solution, the Amount of Chemicals Required and the Rate of Decomposition.—The advantages of keeping the

volume of stock-solution as small as possible are, firstly, less heating to preserve a given temperature, if the solutions are to be used warm; secondly, less decomposition per day or per ton, and consequently less consumption of chemicals. The conditions upon which the volume of the stock-solution depend are: first, the state of the ore, *i. e.*, raw or roasted, raw ore usually requiring less solution than roasted ore; secondly, the capacity of the works per day, the volume of the stock-solution being less in proportion for works of 100 tons capacity than for those of 50 or 25 tons, as already shown;* thirdly, the size of the charges, the volume of stock-solution being less in proportion for 50-ton than for 10-ton charges; fourthly, the regularity of the leaching, irregularity of using the solution necessitating a greater volume.

The strength of stock-solution used at various works for roasted ores is as follows (expressed in percentages): Ontario, 1.4; Daly, 1.4; San Antonio, 0.8; Sombrerete, 1.0; Yedras, 1.8; Lake Valley, 0.8; Chloride, 1.1; San Bartolo, 0.7.

In starting works the stock-solution is generally made up with 1.5 per cent. of hyposulphite, the amount required for 3500 cubic feet being 3281 lbs.; for 2000 feet, 1875 lbs.; and for 1500 cubic feet, 1406 lbs.

The following statement gives the volume of the ordinary or stock-solution used by the Russell Process per ton of ore.

	Volumes of Stock-Solution Cubic Feet.
Silver Reef tailings,	100
Bremen† "	9
Veta Grande "	70
Silver Reef raw ore	137
Sierra Grande roasted ore,	80
Yedras " "	100
Ontario, 1887–8 " "	196
Daly " "	180
Veta Grande " "	110
Ontario, 83–84, " "	190
Sombrerete " "	90
San Antonio " "	100
Chloride " "	102
San Bartolo " "	80
San Miguel " "	70

* For leaching works, as now constructed, the volume of stock-solutions may be put as follows: For 100 tons' capacity, 3500 cubic feet; for 50 tons, 2000 cubic feet; for 25 tons, 1500 cubic feet.

† The small volume of solution used on the Bremen tailings is due largely to its strength (3 per cent.) in hyposulphite. Owing to the extreme slowness of the

The strength of these stock solutions in hyposulphite and the volumes of the first and last wash-water have already been given. The volumes and strengths of the extras are given elsewhere. This table, therefore, completes the description of the liquids used in the Russell process.

For simple alkaline ores, the solutions have all, so far, been used cold, and for raw ores and acid-roasted ores, at a temperature of 110° to 150° F. In the treatment of acid-roasted ores, even if the rest of the stock-solution has been used cold, that portion immediately following the extra-solution, and amounting to 18 to 24 cubic feet per ton, should be warm. In such cases, therefore, the use of an additional storage-tank for hot solution is necessary.

The ordinary or stock-solution is made by dissolving a proper quantity of sodium hyposulphite in a measured volume of water in the storage tanks. Generally the stock solution is made up of a strength of 1½ per cent. hyposulphite (94 lbs. for each 100 cubic feet of water).

After a little practice a stock-solution of one half the strength may often be used. The causes of the deterioration of the stock-solution have already been given. In making up the extra-solution, for each pound of blue-stone used, one pound of hyposulphite of soda is temporarily destroyed, but is regained when the metals are precipitated by sodium sulphide. After the most advantageous concentration of the solution has been determined by experience, it should be maintained by constant addition of sodium hyposulphite: this addition being made to the extra-solution for reasons already given. The strength of the solution in hyposulphite after it has been in use cannot be approximately estimated by its specific gravity for reasons that will soon become apparent. Hence a chemical test is the only one upon which reliance can be placed. The Iodine method has already been given and it is the quickest and most reliable. Another method is to ascertain how much silver chloride will be dissolved by a certain number of cubic centimeters of the solution, but this is somewhat unreliable, except in experienced hands, as two hyposulphites of silver may be formed, one of which forms imme-

leaching, a strong solution was used to shorten the time as much as possible. In nearly all the cases, except that just mentioned, less solution could probably have been used if it had been desirable to do so, but such a course would have made necessary a preliminary assay of the tailings before removal, which would have caused delay. In nearly all cases the tailings are removed before the assay of the tailings is known, a course which is nearly always allowable if the leaching is systematically conducted.

diately, while the other forms only slowly and contains twice as much silver as the other.

The original stock-solution gradually accumulates impurities, such as sodium chloride and sulphate and others, to slight extent, the source of which is evident. The first of these is perfectly harmless, the second diminishes the solvent energy of the solution to a very slight extent, the effect of which is simply to correspondingly increase (by 2 or 3 per cent.) the volume of stock-solution required per ton of ore. Calcium salts are introduced if gypsum is present in the raw ore, or if caustic lime or calcium sulphate exists in roasted ore. In case the lead is precipitated by soda some of the calcium is also precipitated, as it would be difficult to discriminate between the two, even if the action of carbonate of soda on the one did not begin until the other had been entirely precipitated. The concentration of the solution in these salts, however, does not go on indefinitely, but reaches a maximum and then remains stationary, so that a new stock-solution is never required, the same original stock-solution being used year after year, although very little of the original chemicals may remain, on account of the losses and dilution, by the wash-water, which either precedes and follows or merely follows the stock-solution. The amount of impurities in the stock-solution at the Ontario, after two years' use, was such that the addition of barium chloride to 1.000 c. c. of the stock-solution yielded a precipitate weighing 31.4 gr., after drying and 30.2 gr. after treating with HCl and igniting. The specific gravity of the solution was about 4° Beaumé.

There is, however, a change in the condition of the stock-solution that is of great importance, namely, its assuming a caustic reaction. Silver ores only that carry a comparatively large percentage of antimonial and arsenical compounds may be benefited by treatment with a caustic solution. Although the presence of caustic alkali, even in small quantities, is usually very injurious, the remedy is most simple and inexpensive, being accomplished either by the addition of sulphuric acid or blue-stone to the stock-solution. If the former is used the amount required will vary from one third to one pound per ton of ore, and the cost from one to four cents per ton. The effect of the extra-solution in neutralizing caustic impurities has already been described.

One of the causes of the weakening of the stock-solution is the decomposition of hyposulphite by taking up oxygen from the air. At the Ontario in '83–'84 leaching in the mill was twice interrupted

and the rate of decomposition of the mill stock-solution determined. The strength of the solution was in each case 1½ to 2 per cent. In the first case, the solution remained unused in the storage-tank for twenty-six days, during which it decomposed five and a half per cent. In the other case, it remained unused thirty-five days, and the decomposition was 17.4 per cent. In this case, the depth of the solution was about 1¼ times its diameter. The extent of surface exposed in proportion to the depth of the liquid has considerable influence on the rate of decomposition. For instance, the rate of decomposition in volumes whose diameter was seven times the depth was 1.8 times (for sodium hyposulphite) and 2.4 times (for calcium hyposulphite) the rate of decomposition for volumes whose depth and diameter were equal.

b. Volume, Strength, Temperature and Preparation of the Extra-Solution, the Amount of Chemicals Required, and the Efficiency of the Extra-Solution in the Extraction of Silver.

1. *Volume and Depth of the Extra-Solution.*—The volume of the extra-solution and its depth, in proportion to the depth of the ore, varies with the state of the ore, whether raw or roasted, and the method of leaching employed. The volume of the extra-solution varies with the volume required to saturate a ton of ore. This, for raw ores or dry tailings, ranges from 6½ to 9 cubic feet per ton. For roasted ores, except the simple alkaline, the variation is 10 to 14 cubic feet per ton. For raw ores and tailings, if the extra-solution is used on the dry ore, the volume of the extra is the same as the volume to saturate. For all roasted ores, except the simple alkaline, and for raw ores or tailings when the extra is not used first, the volume is 6 to 10 per cent. less than that of saturation, for a reason explained further on. For simple alkaline roasted ore, it is four to six times the volume required to saturate.

The relation of the depth of the extra-solution to the depth of a wet charge, *i.e.*, the depth after leaching has commenced, is as follows: For raw ores and raw tailings, the depth of the extra-solution varies from 36 to 42 per cent. of the depth of the charge of ore; for all roasted ores, except the simple alkaline, it varies from 34 to 46 per cent. of the depth of ore; for simple alkaline roasted ores, it varies from 1½ to 3 times the depth of ore.

2. *Reasons for Adding all the Hyposulphite to the Extra instead of the Ordinary Solution.*—The first reason is that, in treating raw ores, the extra-solution is nearly always used first. As it is

not preceded by any wash-water, but is used on dry ore, no loss of chemicals occurs at this part of the leaching.

Secondly, In treating "simple alkaline" roasted ore, the extraction of a large part of the silver must be done by hyposulphite of soda or lime in the presence of cuprous hyposulphite. But when blue-stone is added to a hyposulphite solution, it absorbs twice its weight of hyposulphite of soda or lime into the cuprous hyposulphite. So, if only that amount of hyposulphite were present, it would all be absorbed in this way, and the extra-solution would fail in this, the most important of its two uses, on simple alkaline ore. The work of the extra, in the use of it here recommended, is more quickly done and at less expense of blue-stone, in proportion (within certain limits) to the strength of the solution in hyposulphite.

Thirdly, When the metals are precipitated from the extra-solution, it becomes converted into ordinary hyposulphite solution, so that the addition of hyposulphite to the extra, is, in all cases, eventually the same as adding it to the ordinary.

Fourthly, As the extra-solution, at the time of making it, must have twice as much hyposulphite as the blue-stone added to it, it is found much more economical to add to the small volume of extra-solution sufficient hyposulphite to raise it to the proper strength (say, 1.75 per cent.) than to maintain the whole volume of stock-solution at that strength.

3. *Strength and Preparation of the Extra-Solutions for Raw, Acid Roasted, and Alkaline Arsenical Roasted Ores.*—The strength of the extra-solutions used on the above ores, varies from 0.7 to 1.1 per cent. in blue-stone, and from 1.5 to 2.3 per cent. in hyposulphite. This percentage of hyposulphite is the percentage existing in the stock-solution at the time of making the extra, plus that added with the blue-stone.

The reason why the extra-solution for roasted ores can be made up on the charge of ore is that it is prevented from sinking into the charge during its preparation by the solution already in the charge, which stands level with the surface of the ore—the outlet of the leaching-tank being stopped while the extra-solution is being prepared. But, in the treatment of raw ores, the extra-solution is generally used on dry ore, which would absorb the extra-solution during the preparation of it, and thus prevent the obtaining of a homogeneous solution.

4. *Strength and Preparation of the Extra-Solution for Simple Alkaline Roasted Ore.*—No rule can be laid down for the preparation

of these extra-solutions, as the volume required may vary from four to six times the amount required to saturate, and the strength from 0.1 to 0.33 per cent. in blue-stone. Besides, for some ores of this class, all or a portion of the blue-stone may be used, as at Lake Valley, in the first wash-water.

5. *Amount of Chemicals Required, and Efficiency of the Extra-Solution in the Extraction of Silver.*—Table XVIII gives the amount of chemicals required for strong, medium and weak extra-solutions, the amount of hyposulphite given in the table varying not only as the amount of blue-stone used, but also as the strength of the stock-solution. The table is for Cusi roasted ore, which has an acid reaction, weighs about 70 lbs. per cubic foot, requires 11 to 12 cubic feet to saturate it, and is treated in charges of about 9 tons. Of course, an ore, requiring only 7 to 10 cubic feet to saturate it, would require less chemicals for the various strengths of stock-solution.

A noticeable fact is the difference in amount of blue-stone and hyposulphite required in the treatment of simple alkaline roasted ore, as compared with acid roasted ore. In Table XIV, the six acid roasted ores require an average of 6.1 lbs. blue-stone, and 4.9 lbs. hyposulphite, while the four simple alkaline ores (Sierra Grande, San Miguel, Veta Grande and Daly) require only 4.5 lbs. of blue-stone and 2.9 lbs. of hyposulphite.

The average maximum extracting-power, in silver per ton of ore of a single medium or strong extra-solution in Table XVIII, containing 6½ to 8 lbs. of blue-stone per ton of ore, is 12.2 oz. per ton. This was determined in twenty-one experiments at Cusi on San Antonio and San Bartolo ore. For these tests charges were selected, having tailings of 20 to 40 oz. per ton after the ordinary solution had done its utmost work and had failed to lower the value of the tailings further. The extra-solution was then used on these charges—sometimes the strong and sometimes the medium of those described in Table XVIII. Of course, on charges yielding tailings of 30 to 40 oz. per ton by the ordinary solution several extra-solutions had to be used to reduce these tailings to 6 or 8 oz. per ton. The results showed that the use of 6½ to 8 lbs. blue-stone per ton increased the average mill-extraction by 12.2 oz. per ton of ore, which is equivalent to 1.68 troy ounces of silver for each avoirdupois pound of blue-stone. At the Ontario, the additional extraction caused by the use of two charges of extra-solution of 9½ lbs. each was an extraction of 39.1 per cent. of 88 oz. ore, more than could be extracted by the ordinary solution in either the mill or the

assay-office. This is an additional extraction of 34.4 oz. per ton or 17.2 oz. for each extra-solution with 9½ lbs. blue-stone, or an extraction of 1.81 oz. of silver per pound of blue-stone. Both the Cusi and Ontario were acid roasted ores.

TABLE XVIII.

TABLE OF CHEMICALS REQUIRED FOR EXTRA-SOLUTIONS OF VARIOUS STRENGTHS FOR CUSI ROASTED ORE: WEIGHTS OF CHEMICALS PER TON AND PER CHARGE OF NINE TONS.

	For a Stock-Solution Containing $\frac{1}{10}$ of One Per cent. Hypo.	For a Stock-Solution Containing $\frac{1}{20}$ of One Per cent. Hypo.	For a Stock-Solution Containing One Per cent. Hypo.	For a Stock-Solution Containing 1.2 Per cent. Hypo.	
	1	2	3	4	
Blue-Stone Per Charge.	46 Lbs.	46 Lbs.	46 Lbs.	46 Lbs.	For a Weak Extra-Solution to contain $\frac{1}{10}$ of One Per cent. in Blue-Stone.
Hyposulphite " "	46 "	33 "	20 "	7 "	
Blue-Stone Per Ton.....	5 "	5 "	5 "	5 "	
Hyposulphite " "	5 "	3¾ "	2½ "	1 "	
	5	6	7	8	
Blue-Stone Per Charge.	59 Lbs.	59 Lbs.	59 Lbs.	59 Lbs.	For a Medium Extra-Solution to contain $\frac{1}{10}$ of One Per cent. in Blue-Stone.
Hyposulphite " "	84 "	72 "	59 "	46 "	
Blue-Stone Per Ton.....	6½ "	6½ "	6½ "	6½ "	
Hyposulphite " "	9¼ "	8 "	6½ "	5 "	
	9	10	11	12	
Blue-Stone Per Charge.	72 Lbs.	72 Lbs.	72 Lbs.	72 Lbs.	For a Strong Extra-Solution to contain 1.1 Per cent. in Blue-Stone.
Hyposulphite " "	111 "	99 "	85 "	72 "	
Blue-Stone Per Ton.....	8 "	8 "	8 "	8 "	
Hyposulphite " "	12 "	11 "	9 "	8 "	

Only for a Roasted Ore which requires twelve cubic feet to saturate.
For Cost of these various Extra-Solutions see Table xxiv.

At the Ontario, when allowed to act on tailings from the ordinary solution, but very high in value, or in other words when allowed to show its utmost efficiency, the amount of silver extracted by the extra-solution was in value about fifteen times its entire cost, including chemicals, preparation and application, and the precipitation of the metals from it. When used on simple alkaline roasted ore, the extra-solution may have a still greater efficiency in extracting silver, due to its double action hereafter referred to.

6. *General Rules to be Observed in the Preparation and Use of the Extra-Solution.*—If the extra-solution is made at a temperature of over 120°, it should not be prepared until just before using. Even cold extra-solutions should not be allowed to stand unused more than six or seven hours. For all ores except the simple alkaline roasted ores, the total weight of hyposulphite used, including that existing in the stock-solution, should be at least twice the weight of blue-stone. For simple alkaline roasted ore, the weight of hyposulphite used, including that existing in the stock-solution, should be at least three times that of the blue-stone.

As already stated, the extra solution is made up either in a separate tank, or on the surface of the charge. The first method is always used for raw ores, and for roasted ores also, if the leaching works are to be run to their utmost capacity. In this method about one-third of the total volume of solution required for the extra-solution is run into the storage tank in which the extra is to be prepared. The proper quantity of chemicals is then placed in the chemical box, and the remaining two-thirds of the stock-solution allowed to run in through this box. If this is not sufficient to dissolve all the chemicals, some of the extra-solution in the tank is circulated through the box by means of a No. 4 syphon pump until they are dissolved. The reason for running one-third of the stock-solution into the tank before dissolving the chemicals is to avoid the formation of the yellow precipitate (cuprous hyposulphite), at the bottom of the tank, which would be dissolved only after considerable stirring. In any case, however, the extra-solution is stirred just before it is used. In making up the extra-solution on the surface of the ore, for all except simple alkaline ores, the chemicals are placed in the chemical box and the whole volume of stock-solution required allowed to pass through them. The extra-solution is stirred just before it is used, and whatever yellow precipitate may be left on the ore will be dissolved during the circulation. But in making up the extra-solution for simple alkaline ores, none of this precipitate forms, since it does not appear if, as in the treatment of them, the amount of blue-stone is considerably less than one-half the amount of hyposulphite. In making up the extra-solution for these ores, the manipulation is the same, except that four to six extras are made up, each containing much less chemicals, which are not circulated, and which are applied directly after the first wash-water, instead of after the ordinary or stock-solution, and cold instead of warm. Practical experience has demonstrated that if roasted ores containing

an appreciable percentage of caustic lime are treated with ordinary solutions after the first wash-water, as must be done if the old process of leaching is used, the result may be very disastrous, and that the damage done is only partially made good by the subsequent leaching with the extra-solution. But if the extra-solution is applied at once, the cuprous hyposulphite neutralizes and counteracts the deleterious effect of the caustic lime, and leaves the silver in a soluble form. Before the extra-solution follows the first wash-water, it has been found beneficial to impregnate the ore with a small quantity of a strong copper sulphate solution. It can easily be seen that the first portion of the extra-solution, replacing the wash-water, must become more or less diluted by contact with water. This dilution may be so considerable that an insufficient quantity of copper is present to protect the silver from the deleterious influence of the caustic lime. Therefore, after the leaching with water is finished, and just before the extra-solution is to be used, about one or two pounds of blue stone per ton of ore is dissolved in an amount of water equal to from one to three inches on the surface of the ore. As soon as this sinks to the surface of the ore, the extra-solution is used. But for some ores all the blue-stone may be used in the first wash-water. This may be done when the ore contains such a compound (usually carbonate of an alkali or alkali earth) as will precipitate all or a portion of the copper out of the wash-water, in the ore, in such a state that it will be wholly or partially dissolved by the ordinary or stock-solution afterwards used, thus making the extra-solution in the ore. The blue-stone, when thus used, is added to about one charge, (or volume to saturate) of wash-water, which may be either passed through the charge only once or circulated. If desirable, it may be followed by more wash-water, in order to wash out any copper not precipitated in the ore.

c. Differences Between the Solutions.

1. *Difference Between the Ordinary and the Extra-Solution and Between the Extra-Solutions.*—No extra-solutions are kept in stock. They are made up fresh for each charge. The extra-solution for simple alkaline ores is used like the ordinary, *i.e.* without circulating, but differs from it, in containing a fraction of 1 per cent. of cuprous hyposulphite, and in losing its efficiency as an extra when precipitated. For all other ores, both raw and roasted, the extra-solution differs from the ordinary in containing from six-tenths of 1 per cent. to 1.2 per cent. of cuprous hyposulphite, in being much less in vol-

ume, and in being circulated, and also in losing its efficiency as an extra when precipitated.

The extra-solutions differ from each other as follows: For simple alkaline roasted ore, the extra is used cold, weak, in comparatively large volume, is not circulated, and is used immediately after the first wash-water; or part or all of the blue-stone may be used in the first wash-water. For all other ores, both raw and roasted, the extra-solution is generally used stronger, warm, as small in volume as possible, after the ordinary solution, is circulated, is followed by hot ordinary, and no blue-stone is used in the first wash-water.

2. *Reasons for the above Differences.*—In the treatment of simple alkaline roasted ore, the extra-solution has a double use; First, in dissolving and extracting the silver and gold compounds, not soluble in the ordinary solution, and secondly, in protecting the silver (and gold) from the injurious effect of the caustic alkali in the ore, while they are being dissolved by the hyposulphite and are passing out of the charge. The first is the regular use of the extra-solution, but in its second use it is of much more value. In this use of the extra, the extraction of that part of the silver soluble in hyposulphite only, is done under the protection, so to speak, of the weak extra-solution. Hence the blue-stone must be used at the beginning, and its use continued until, as shown by experience, three to six charges of solution have been used. Its use is then no longer necessary. It should be remembered that a "charge of solution" is an amount of solution equal to the amount required to saturate the "charge of ore."

3. *Reasons for Circulating and for Small Volume of Extra.*—For those ores on which the extra is circulated, it must be of a certain strength, say, 0.6 per cent., to accomplish its object. Therefore, the smaller the volume of extra per ton, the less will be the cost of chemicals per ton. The reasons for circulating the extra but not the ordinary, are two: in the first place, the ordinary solution, in passing through the precipitating-tanks, renews its strength as an ordinary, and is ready to be used again. There is, therefore, no reason why it should be circulated before being precipitated. The extra-solution, on the contrary, in passing through the precipitating-tanks, loses its efficiency as an extra, being converted into ordinary solution. Therefore, the whole effect of the extra should be obtained before allowing it to be precipitated, which can only be done by circulating it. In the second place, the volume of the extra is the smallest possible that will saturate the ore, and, on this account, in passing through the charge only once, all particles of the ore might

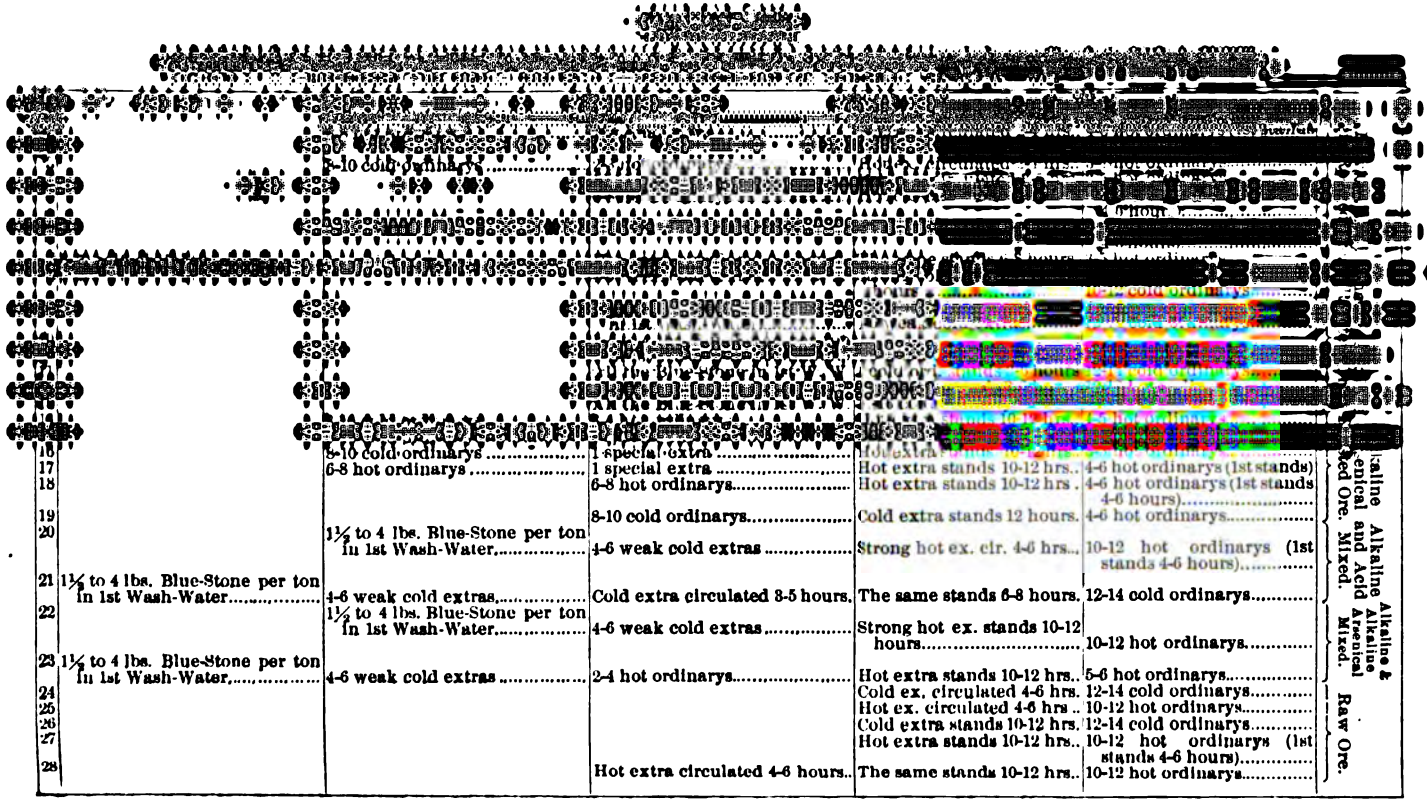
not be brought in contact with it. On the other hand, the volume of the ordinary solution is practically unlimited, the storage-tanks being always kept supplied from the precipitating-tanks at no additional cost for chemicals.

d. Manipulation of the Extra- and Special Extra-Solutions.—When the extra-solution is to be circulated, the outlet leading to the precipitating tanks is stopped when the surface of the extra-solution has sunk to one-half inch above the surface of the ore. Steam is then turned into the syphon-pump, and the solution returned on to the surface of the ore continuously from four to six hours. For simple alkaline roasted ore the extra is allowed to pass through the ore as if it were the ordinary.

The extra-solution for raw and for acid roasted ore should be followed by hot ordinary. Also, in the treatment of simple alkaline ores, all the blue-stone may sometimes be used in the first wash-water by adding 2 to 4 pounds of blue-stone per ton of ore to about one-half or two-thirds of a charge of wash-water, after the washing is about finished, and allowing it to pass through the ore like the rest of the wash-water. It may sometimes be followed by more wash-water; but usually it immediately precedes the ordinary. In the treatment of alkaline arsenical ores, ordinary solution is used before the extra, until nearly all the silver has been extracted which is soluble in the ordinary solution. The extra-solution is then used, being always allowed to stand from 6 to 12 hours in the charge, although it may then be circulated. It may be used cold or hot, but is always followed by hot ordinary, when used cold. The formation of a yellow precipitate, during the preparation of the extra-solution, which fails to dissolve by the time the extra is ready for use, indicates an insufficient amount of hyposulphite.

Acid ores and simple alkaline ores (according to experience so far) should not be mixed in the same charge. In case such a mixture should occur, the charge should be treated first as if it were alkaline ore, and then as if it were acid.

The special extra-solution is merely an extra-solution which has already been used upon one charge. When used upon a second charge, it is always applied immediately before the regular extra. The main object in using it is, that the regular extra-solution may be diluted by a solution of its own kind, as it is sure to become diluted to some extent with whatever solution is in the charge. Another reason is, that the efficiency of an extra-solution is not usually exhausted by using it on one charge. Its use saves chem-



PROCESS.

16		8-10 cold ordinarys	1 special extra	Hot extra stands 10-12 hrs.	4-6 hot ordinarys (1st stands)	Alkaline Arsenical and Acid Mixed.
17		6-8 hot ordinarys	1 special extra	Hot extra stands 10-12 hrs.	4-6 hot ordinarys (1st stands 4-6 hours)	
18			6-8 hot ordinarys	Hot extra stands 10-12 hrs.	4-6 hot ordinarys	
19			8-10 cold ordinarys	Cold extra stands 12 hours.	4-6 hot ordinarys	
20		1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	Strong hot ex. cir. 4-6 hrs.	10-12 hot ordinarys (1st stands 4-6 hours)	
21	1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	Cold extra circulated 3-5 hours.	The same stands 6-8 hours.	12-14 cold ordinarys	
22		1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	Strong hot ex. stands 10-12 hours	10-12 hot ordinarys	Alkaline & Alkaline Arsenical Mixed.
23	1½ to 4 lbs. Blue-Stone per ton in 1st Wash-Water	4-6 weak cold extras	2-4 hot ordinarys	Hot extra stands 10-12 hrs.	5-6 hot ordinarys	
24				Cold ex. circulated 4-6 hrs.	12-14 cold ordinarys	Raw Ore.
25				Hot ex. circulated 4-6 hrs.	10-12 hot ordinarys	
26				Cold extra stands 10-12 hrs.	12-14 cold ordinarys	
27				Hot extra stands 10-12 hrs.	10-12 hot ordinarys (1st stands 4-6 hours)	
28			Hot extra circulated 4-6 hours.	The same stands 10-12 hrs.	10-12 hot ordinarys	

icals and causes a higher extraction of the silver, and, for the same amount of blue-stone per ton, the product is of a higher grade if the extra-solution is used as a "special extra."

Also the use of the "special extra" secures a more even extraction of the silver from all parts of the charge of ore. For instance, at Cusi, on San Bartolo ore, the tailings from the lowest six inches of the tub averaged 7.0 oz. On the other hand, while using the special extra, similar samples averaged 5.3 oz.

It may be used on any ore, in the treatment of which the extra-solution would follow the ordinary. It is not circulated, but precedes the extra as if it were ordinary solution, and passes from the charge on which it has been used directly to the precipitating-tanks.

e. Order of Application, Volumes and Temperatures of the Leaching-Solutions.—Table XIX gives the order of applying the leaching-solutions on each of the different classes of ore. The first and second wash-waters are omitted from the table.

The "ordinarys" and "extras" referred to are each "charges" of those solutions, the definition of which term has already been given. The volumes of solutions given have proved sufficient for all ores so far treated by the Russell process.

After once selecting the method which gives the best result for the ore from any particular mine, no change in method will usually be required for the ore from that mine or district, unless the wash-water from these ores changes from alkaline to acid or the reverse. For custom-mills treating several different kinds of ore, both acid and alkaline, each one of two or even three methods may have to be used. According to some of the methods given in Table XIX the extra-solution is *required* to stand in the ore, while in others it is unnecessary, but, so far, no injurious effects have been observed due to the standing of the extra-solution *in* any kind of ore. But it should not be allowed to stand more than a few hours, except in the ore.

Table XX gives the results of treating an acid roasted ore as if it were alkaline-arsenical. At the time of making this comparison, some of the leaching-tanks at the Cusi mill had not been fitted with syphon-pumps or ejectors with which to circulate the extra-solution. It was therefore necessary to obtain the best results possible by allowing the extra to stand in the ore. The average difference in mill-extraction by the two methods was 6.9 per cent.

TABLE XX.

EFFECT OF CIRCULATING THE EXTRA-SOLUTION AS COMPARED WITH NOT CIRCULATING (ACID-ROASTED ORE) AT CUSI MILL ON CUSI ORE.

"Circulated" or "Not Circulated."	Number of Charges compared.	Value of Ore in oz. Silver Per Ton.	Per cent. Extraction by Ordin'ry in Ass'y Office.	Per cent. Extraction by Extra in Ass'y Office.	Per cent. Extraction in Mill.
No Circulation. The extra stood in the ore for twelve hours.	46	34.8	77.1	89.0	78.2
Circulation. The extra was circulated for four hours.	96	34.8	76.6	87.6	85.1

Note.—The above is a roasted ore with acid first wash-water.

Table XXI, on the other hand, shows the effect on an alkaline-arsenical ore of circulating the extra instead of allowing it to stand, the difference in mill extraction by the two methods being 4.9 per cent. of the value of the ore.

TABLE XXI.

EFFECT OF ALLOWING THE EXTRA TO STAND TWELVE HOURS IN THE ORE WITHOUT CIRCULATION AS COMPARED WITH CIRCULATING THE EXTRA (ALKALINE-ARSENICAL ROASTED ORE) AT YEDRAS MILL ON YEDRAS ORE.

"Circulated" or "Allowed to Stand Twelve Hours without Circulating."	Value of Ore in oz. Silver Per Ton.	Per cent. Extraction by Ordinary in Assay Office.	Per cent. Extraction by Extra in Assay Office.	Per cent. Extraction in Mill.
Circulation. The extra solution was circulated for four hours.	57.4	77.0	83.3	77.7
No Circulation. The extra solution stood in the ore for twelve hours.	63.5	70.6	83.1	82.6

Note.—The above is a roasted ore with an alkaline wash-water, but with much arsenic in the ore.

F. *Preparation and Manipulation of the Precipitants, Sodium Sulphide and Soda-Ash, and the Amount of Each Used.*

a. Preparation of the Sodium Carbonate (or Soda-Ash) Solution.—In the preparation of the sodium carbonate solution, if the ammonia or Solvay process soda-ash is used, it is dissolved in an iron tank,

in a portion of the hyposulphite stock-solution, in preference to water, so that, in precipitating the lead, the strength of the stock-solution is not diminished by dilution. If an aqueous solution of the soda-ash be used, it should be made as strong as possible, and kept heated. If ammonia process soda-ash cannot be obtained, and the ash used contains, as is usually the case, some sodium sulphide, the removal of this sodium sulphide becomes necessary for the following reasons: The sulphur of the sodium sulphide, having a greater affinity for silver than for lead, will precipitate silver out of the leaching-solution when the soda-ash is used to precipitate the lead. Consequently, the precipitate of carbonate of lead will be contaminated with sulphide of silver to the extent of about 100 ounces of silver per ton of lead carbonate, instead of 1.5 ounces per ton as would be the case if the soda-ash were free from sodium sulphide. The removal of this sodium sulphide from the soda-ash, is, however, a very simple matter. As carbonate of copper is perfectly soluble in a hyposulphite solution, while sulphide of copper is not, it is evident that, if the impure soda-ash is dissolved in a hyposulphite solution (such as the stock-solution), and then an amount of blue-stone (or other compound of copper soluble in the hyposulphite solution) corresponding to the amount of sodium sulphide be added to the hyposulphite solution of soda-ash, all the sodium sulphide will be neutralized by the formation of sulphide of copper which sinks to the bottom as a black precipitate. The clear solution is then drawn off and is ready for use. Generally about one pound of copper sulphate is sufficient to purify 100 pounds of soda-ash. If caustic soda is present in the soda-ash, it may be removed by boiling it with a little sulphur, and then adding copper sulphate. The presence of sodium sulphide in the ash may be detected by adding a little of an aqueous solution of the ash to a hyposulphite solution containing silver, a dark discoloration or the formation of precipitate indicating the presence of sodium sulphide.

b. Preparation of the Sodium Sulphide Solution.—In order to obtain a sodium sulphide of maximum precipitating power, the following method should be strictly followed: The whole contents of a drum of caustic soda, 600 to 700 pounds, are broken up into lumps which should not exceed 6 to 8 pounds in weight, and placed in the cast-iron tank with false bottom of lead, hereafter described. About 2 or 3 cubic feet of water are added, and dry steam turned on as full as possible, the tank being covered to avoid spattering of the lye. If sufficient head of steam is used, the dissolving of the caustic soda

will be accomplished in ½ to ¾ hours. It would not do to attempt to dissolve it slowly, since too much steam would be condensed. The total volume when the mass is dissolved should be such as to measure not more than as many cubic feet as there were pounds of caustic soda divided by 62.5. In other words, one cubic foot of lye should hold at least 62.5 pounds of caustic soda. As soon as the caustic soda is dissolved, and while the lye is at a temperature of not less than 200° F., which will be the case if the above directions are followed, the sulphur is added, a shovelful at a time. If it is lump-sulphur, it is not necessary to crush the lumps unless most of them will not pass a screen of 1-inch mesh. The sulphur must be added slowly, as the mass boils violently, and swells to several times its original volume while the chemical action is taking place. Should the lye have been either much less concentrated or of lower temperature than stated above, the solution of the sulphur will not be complete.

The sodium sulphide should be made in a cast-iron tank of not more than eight or nine feet area. If through mismanagement the lye is not as concentrated or as hot as above described, and the mass does not boil violently when the sulphur is added, it (with the sulphur) must be boiled with steam for about two hours. The precipitating-coefficient of the finished product is, however, thereby diminished. The sodium sulphide solution obtained as above described is so concentrated, that it would solidify on cooling. It is therefore diluted with hyposulphite stock-solution in preference to water, after being transferred to the storage-tanks by means of a short piece of wooden trough not shown in the drawings. The dilution is carried to such an extent that the total volume measures as many cubic feet as there have been used pounds of caustic soda, divided by 12. If the sodium sulphide still shows a tendency to crystallize, the dilution may be carried still further. The silver-precipitating coefficients for caustic soda and sulphur thus prepared are about 2 to 2.3 for silver and 3.5 to 4 for sulphur.

After the sodium sulphide in the cast-iron tank has ceased boiling, it may be so thick that it will not easily run out, in which case it is diluted by solution or water in the cast-iron tank to the extent of about its own volume.

c. Manipulation of the Sodium Carbonate (or Soda-Ash) Solution.—The precipitation of lead with sodium carbonate is very easily effected, and the precipitate settles quickly, leaving the solution clear. The precipitate is also more compact than the sul-

phide. Care should be taken not to add an excess of soda (although the presence of this reagent in the hyposulphite solution is not in the least injurious to the extraction of silver in lixiviation, even if the solution was saturated with it), as it can easily be seen, that, should it become necessary to neutralize any caustic soda in the solution, the consumption of sulphuric acid would be increased by neutralizing sodium carbonate also.

The end of the reaction, or the time at which the addition of the sodium carbonate should be stopped, is best determined by taking some of the solution from the precipitating-tanks in a test-tube, and adding a few drops of an aqueous solution of sodium phosphate, which gives a more delicate reaction than sodium carbonate. After the lead carbonate has settled in the precipitating-tanks, which occupies about half an hour, the clear solution is decanted or pumped by an ejector into the silver-precipitating-tanks. After the final addition of soda-ash solution, the whole contents of the precipitating-tanks are stirred with a hard-wood oar for about half a minute. Also, during the addition of the soda-ash the solution should be stirred. The lead carbonate is removed about once a week, and pressed in a filter-press. The lead carbonate is very pure, unless lime is present in the solution, in which case it will be contaminated to some extent with the lime. In practice, about one pound of soda is consumed in precipitating one pound of lead, or one and one-third pounds of lead carbonate.

d. Manipulation of the Sodium Sulphide Solution.—During the addition of sodium sulphide to the leaching solution, in the precipitating-tanks, and at the end of the operation the solution should be well stirred. The more violent the stirring, the quicker and more compact will be the settling of the precipitate and the clearer the solution. It is hardly necessary to say that the sodium sulphide should not be used in excess. In case this has been done by inadvertence, more fresh silver-bearing solution must be added. Hence, in filling the precipitating-tanks, space should be left for the addition of a few more inches of solution if necessary. The addition of an excess of sodium sulphide is indicated by a smell of hydrogen sulphide, a turbidity of the solution, and a discoloration of the sides of the tank, or the floor around it. If it should happen that an over-precipitated solution has been run on a charge of ore, it is only necessary to follow it with an extra-solution which will dissolve the sulphide of silver precipitated in the ore.

If the solution to be precipitated contains lime, an insoluble cal-

ed, and may not only contami-
eatly increase the amount of
f doubling it. But the light
pear until all the sulphides of
t. In order to determine when
ne of the solution is taken in
olution is added to it in excess.
ve the copper and silver, if any,
the test-tube is then decanted
sulphide is added to it. The
ellow calcium monosulphide,
ipitate will show that all the
pitated. After the sulphides
nted by means of the surface-

XII.

D COPPER, IN THE HYPOSULPHITE
RY LEACHING PROCESS AT
ONTARIO ORE).

c.c. used for the 8th Twelfth.	c.c. used for the 9th Twelfth.	c. c. used for the 10th Twelfth.	c. c. used for the 11th Twelfth.	c. c. used for the 12th Twelfth.	
326	335	374	390	380	Soda-Ash Precipitating-Solution.
90	64	45	41	33	Sodium Sulphide Precipitating-Solution.

ows the comparative amounts
uired at different stages of the
distribution of the lead, silver,
n treating Ontario roasted ore.
d were of small size, holding
e also two of the same size for
Each tank was filled and
he number of c.c. of soda-ash

solution required to precipitate all the lead from each one of the tanks, each time, was noted, and also the number of c.c. of sodium sulphide used each time to precipitate the silver and copper. The Table shows that the amount of lead dissolved by a given volume of solution remained about the same at different periods of the leaching. But the amount of silver and copper dissolved by the second one-twelfth of the solution was nearly double that dissolved by any other one-twelfth, and the amount dissolved by the first one-sixth was 44.7 per cent. of the whole.

In these experiments the leaching with solution was commenced about the middle of the day, stopped during the night, and continued until about 11 A.M. the next day. During the night the solution remained standing in the ore. But the solution which had remained in the ore all night contained no more metal than if it had passed rapidly through. The strength of sodium sulphide solution was much weaker than now used. The strength of the cold stock-solution used was 0.8 to 2.0 per cent., the value of ore 70 to 110 oz., the depth of the charge when dry about 22 inches, and after leaching 16.5 inches. The extra-solution was not used until after the twelfth charge. It was noticed that the volume of solution containing the extra-solution contained nearly as much lead as any other volume. The carbonate of lead produced was pure white, perfectly free from any other metal, except that it contained about 1.5 oz. of silver per ton of lead carbonate. The value of the dried sulphides was from 7500 to 12,000 oz. per ton.

TABLE XXIII.

VALUE OF THE PRECIPITATE OR "SULPHIDES" FROM THE LEACHING-SOLUTIONS AND THE COST OF PRECIPITATING THE SAME WHILE USING THE RUSSELL PROCESS.

Name of Mine or Mill.	Ore or Tailings.	Sulphides "Roasted" or "Dried."	Ounces Silver per ton in the Sulphides.	Ounces Gold per ton in the Sulphides.	Cost of Precipitating per ton of Ore.
Leeds	Tailings.	Dried.	3000	Trace.	34 cents.
Bremen	"	Roasted.	8000	None.	14 "
R. & E. & M. V.	"	Dried.	.....	.....	17 "
Veta Grande	"	Roasted.	5400	20.	30 "
Silver Reef	Raw Ore.	Dried.	3300	Trace.	38 "
Raymond & Ely	"	Dried.	11600	13.2	20 "
Sierra Grande	Roasted Ore.	Roasted.	8040	None.	11 "
San Miguel	"	Dried.	8980	3.2	47 "
Yedras	"	Roasted.	15000	Not Assayed.	86 "
Veta Grande	"	Roasted.	8400	39.0	54 "
Sierra Grande	Roasted Tailings.	Roasted.	7200	None.	11 "
Ontario	Roasted Ore.	Dried.	10500	Not Assayed.	39 "
Sombrerete	"	Roasted.	7400	36.0	55 "
San Antonio	"	Dried.	8200	4.3	88 "
Chloride	"	.....	.....	.....	41 "
San Miguel	"	Dried.	7000	2.8	64 "
San Bartolo	"	Roasted.	8790	8.5	58 "

Table XXIII gives the value in silver and gold of the precipitates or sulphides produced by the Russell process in various mills. The sulphides produced from the Leeds and other Silver Reef ores are low in grade, on account of the presence in the ore of carbonate of copper which is dissolved by the hyposulphite solution.

G.—*Mechanical Treatment of the Products.*

The precipitated sulphides are removed from the three precipitating-tanks for solution about once in three days, and the precipitate from the wash-water tanks at the same time if the wash-water has been precipitated by sodium sulphide, otherwise, usually only once a week. To remove the sulphides, the clear solution is decanted by means of the surface discharge as low as possible. The gate or valve on the rear of the precipitating-tanks is then opened, and the sulphides allowed to run out into the trough or launder leading to the tank for the storage of sulphides. The sulphides are permitted to settle in this tank, and the clear solution again decanted, so as to save running it through the press. The storage-tank, of the size given in the drawing, will hold the sulphides for one week's run, or about 700 tons of average ore. The sulphides may be made more compact and reduced in bulk about two-thirds by using a jet of steam in the storage-tank. But, if this is done, the sulphides should be allowed to cool before pressing, for reasons explained further on. When the sulphides are to be pressed, all the valves between the storage-tank for sulphides and the filter-press (except the steam valves on top of the pressure-tank) are opened, so that the sulphides run freely through the pressure-tank into the press. In this way, two-thirds of the pressing is accomplished by gravity. While the sulphides are running into the press, the contents of the tank are occasionally stirred. When the press is so full that the solution comes only slowly from it, the valve between the sulphide storage-tank and the iron press-tank is closed, and as soon as the liquid in the pressure-tank has sunk below the level of the small valve on the side of the pressure-tank, three or four inches below the top, steam at full pressure is turned into the pressure-tank, until again the clear solution comes only slowly from the filter-press. It is best not to have the pressure-tank quite full when the steam is turned into it, as the solution would be heated by it, and a hot solution has a bad effect on the filter cloths, so that they do not last more than one-half as long as they otherwise would. When the press is full, which

is indicated by the pressure gauge, as well as by the slow running of the clear solution from the press, the steam is turned off and allowed to escape from the pressure-tank, and the valve between the pressure-tank and the press is closed. The trough under the press, for conducting off the clear solution, is then removed, and a large iron pan about three feet wide, five inches deep and long enough to reach the whole length of the press between the legs, and having handles, is placed underneath the press. The press is then emptied by sliding the plates apart, and inserting a wooden (not a metal) paddle behind the cakes which causes them to drop into the pan. If distance-rings are used, they are removed from press by the handles and the cakes pushed out. When the press is first opened, a small amount of solution may run out, so that it is best to have the press set in a low tank, about two inches deep, and with a drain-pipe in one corner leading down to the sump under the press.

An 18-inch press has a capacity without distance-rings corresponding to about 250 pounds of dry sulphides per hour. The distance-rings are used when a greater capacity is desired, but the cakes will then contain a greater percentage of moisture.

The appearance of sulphides pressed at 150 pounds per inch does not indicate the presence of more than 10 per cent. moisture, but they have in reality about 38 to 42 per cent. If these sulphides are dried in a reverberatory furnace, 0.5 per cent is likely to be lost in dust, and considerable labor is involved.

The best method of drying is by steam, in which there is no loss, and which requires only a nominal amount of labor. After being dried, which occupies two or three days, the sulphides are sampled on the iron floor beside the steam dryer. The lead carbonate precipitate is pressed and dried in the same manner as the sulphides.

H.—*General Remarks.*

While the first wash-water is draining from the leaching-tank, preparatory to using the leaching-solutions, cracks are likely to be formed, due to a slight settling of the ore. Before the leaching-solutions are used these cracks should be filled up by smoothing over the surface of the charge with a wooden hoe. Also before and during the use of the extra-solution, particular care should be used in filling up any cracks, especially around the sides of the charge; otherwise the effect of the extra-solution may be decreased.

For maintaining a partial vacuum under the filter and thus acceler-

ating the leaching and circulating the extra-solution, the Aller lead-lined siphon-pump is the best. It is applicable in all cases except on such extremely fine material as that at Silver City, another case of which is not likely to be found. The various sizes used are Nos. 2, 3 and 4 for leaching-tanks of 12, 14 and 16 feet diameter, respectively. If the leaching-rate without a vacuum is 6 to 8 inches an hour, the use of a siphon-pump will usually double it. If only one-quarter to one-half an inch per hour, the use of a siphon-pump will often raise it to 6 or 8 inches. If a more positive vacuum is necessary, as was the case at Silver City, a No. 4 or 5 Knowles' pump is used, one being sufficient for a mill of 50 tons' capacity.

The lowest six inches of the charge of ore will sometimes show a less thorough extraction of the silver than the uppermost six inches, or than the average of the whole charge. Therefore, in starting new works, or if the value of the tailings is not satisfactory, the lowest six inches of the tailings should be sampled, and this should also be done whenever two methods of leaching are to be compared. If the tailings are sluiced out the amount of water required for sluicing is twelve to eighteen feet per ton of ore. But if roasted ore is being treated no additional water is required, as the last half of the first wash-water can be used for this purpose, being conducted down to and stored in a suitable tank near the fire-pump.

III. Leaching Plant.

A.—*General Remarks.*

In Mr. Stetefeldt's paper on the Russell process in its practical application (*Transactions*, vol. xv.) a short description was given of the plant required for the Russell process. This, however, in view of the many improvements made in the year and a half since the date of that paper is not sufficient, especially as no plans and only one detailed drawing were given. The working drawings here given comprise the best features of five leaching plants, built especially for the process, and of the Sombrerete plant now being constructed. With the exception of two minor details on the precipitating-tanks there is nothing given but what has stood the test of actual practical experience. The plant is entirely independent and complete as it stands and may be situated wherever convenient, without refer-

ence to the position of the crushing plant for raw ores, or for the crushing chlorodizing plant for roasting ores, except as regards the easy transportation of the raw or roasted ore between the two plants.

Consequently when either one of the two above-mentioned plants for the preparation of the ore already exists in connection with amalgamation, the change to the Russell process can be made by the erection of the plant herein described without stopping or removing the amalgamation plant. The plant has its own boiler, and no engine is required.

The following is an index to the parts of the plant as given in Plates I and II. The details are given in Figs. 1 to 3, and Figs. 6 to 19.

Index to Parts of Plant.

A. Leaching-tanks.
B. Solution-precipitating-tank.
C. Wash-water.
D. Solution-pump-sump.
E. Storage-tank for sulphides.
F. Filter-press-sump.
G. Storage-tanks for solution.
H. Storage-tank for hot solution, or for preparation of extra.
I. Cast-iron tank for preparation of sodium sulphide.
J. Sheet-iron tanks for storing sodium sulphide.
K. Iron pressure-tank.
L. Filter-press for sulphides.
M. Steam-dryer for sulphides.
N. Boiler.
O. Fire pump for sluicing.
P. Solution-pump.
Q. Main ore-track to leaching tanks.
R. Side-tracks to leaching-tanks.
S. Side-tracks in chemical room.
T. Grade-line between boiler and steam-dryer.
U. Grade-line through sulphide storage-tank.

a. Tailing-shute (to cars).
b. Sluice-gate for tailings.
c. Sluice-trough for tailings.
d. Box for chemicals for extra-solution.
e. Ore-car.
f. Ore-car truck.
g. Chemical car.
h. Special-extra trough over leaching-tanks.
i. Solution-trough over leaching-tanks.
j. Solution-trough to solution precipitating-tanks.
k. Wash-water trough to wash-water precipitating-tanks.
l. Waste-water trough.
m. Trough to conduct sulphides to sulphide storage-tank.
n. Trough to conduct solution to solution-pump-sump.
o. Waste wash-water trough.
p. Solution-pipe to leaching-tanks.
q. Hot-solution or extra-solution pipe to leaching-tanks.
r. Water-pipe to leaching-tanks.
s. Hose for circulating extra.
t. Hose from leaching-tank to siphon-pump.
u. Hose from leaching-tank to triple launder.
v. Hose from siphon-pump to triple launder.
w. Siphon-pump.
x. Tailing-car.
y. Hose-connections for sluicing.
z. Waste-water pipe from triple launder.

a'. Pipes from triple launder to solution-precipitating-tanks.
b'. Pipes from triple launder to wash-water precipitating-tanks.
c'. Valve or "gate" for sulphides.
d'. Surface-discharge for precipitating-tanks.
e'. Sodium-sulphide pipe.
f'. Pipe for precipitated wash-water.
g'. Connection between solution-pump and sump.
h'. Box and strainer around outlet.
i'. Drip-pipe from solution-pump.
j'. Pipe from pump to storage-tanks.
k'. Connection between sulphide-storage-tank and pressure-tank.
l'. Connection between pressure-tank and press.
m'. Siphon-pump in filter-press sump.
n'. Pipe for returning solution to solution-sump.
o'. Small windlass for steam-dryer.
p'. Boxes over storage-tanks to prevent agitation of the solution.
q'. Boxes and strainers around outlets.
r'. Lead heating-coils in storage-tanks.
s'. Drip from lead coils.
t'. Triple connection between the three storage-tanks.
u'. Main steam-pipe.
v'. Water-pipe from fire-pump for sluicing.
w'. Storage-room for caustic soda and sulphur.
x'. Storage-room for blue-stone and hyposulphite.
y'. Trap-door to filter-press sump.
z'. Gauge-valve on side of pressure-tank.

a''. Solution-pipe to sodium sulphide storage-tanks.
b''. Pipe for drawing off sediment from pump-sump.
c''. Pipe for drawing off sediment from solution storage-tanks.
d''. Siphon-pump for circulating solution through chemical-box in extra solution-tank.

TABLE XXIIIa.

CAPACITY OF THE PLANT PER DAY FOR VARIOUS ORES AND TAILINGS TREATED BY THE RUSSELL PROCESS.

Name of Ore.	Kind of Material.	Weight of One Cubic Foot in Pounds.	Depth of Charge if Extra Solution is made up in Separate Tank.	Corresponding Weight of Charge in Tons.	Total Capacity of Plant per day if Tailings are Sluiced Out.	Total Capacity of Plant per day if Tailings are Shoveled out.
Silver Reef	Raw ore and tailings.	80.	Full (= 82 inches.)	55.	110 Tons.	100 Tons.
Pioche		100.		68.	155 "	135 "
Parral		80.		55.	110 "	100 "
Lake Valley	Roasted ore.	86.		59.	170 "	140 "
Parral		88.		60.	145 "	120 "
Yedras		60.		41.	80 "	70 "
Ontario		65.		44.	130 "	110 "
Daly		65.		44.	130 "	110 "
Sombrerete		60.		41.	120 "	100 "
Cusi		70.		48.	110 "	100 "
Chloride		79.		54.	140 "	120 "

12 hours allowed for filling each tank.
2 " " " emptying " " if the tailings are "sluiced out."
12 " " " " " " " " " " "shoveled out."

Table XXIIIa gives the capacity of the plant for various raw and roasted ores and tailings in tons per day. This capacity may be further increased to the extent of 30 per cent. by the addition of two more leaching-tanks, without making any change in the rest of the plant. On the other hand, it is not advisable to erect less than six leaching-tanks, since the remainder of the plant should be the same even if only four leaching-tanks were used. In the drawings showing the elevation of the plant, imaginary grade-lines are shown, one being about 25° and the other about 30°. These grades would be the most suitable, but they are by no means necessary. At Sombrerete the same kind of plant is being constructed on a grade of only 12° to 15°.

The building containing the pressure-tank, filter-press, and steam-dryer can be put in front of the sump for the solution-pump and the storage-tank for sulphides, instead of at the side.

In the plan, the floors are supposed to be removed, but are shown in the elevation. Pipes, details and dimensions are omitted in some of the views, in order to avoid a complicated drawing, but are given in the others. Figures alongside of pipe-lines indicate the sizes of the pipes. All tanks of all kinds, and all troughs or launders are accessible on every side.

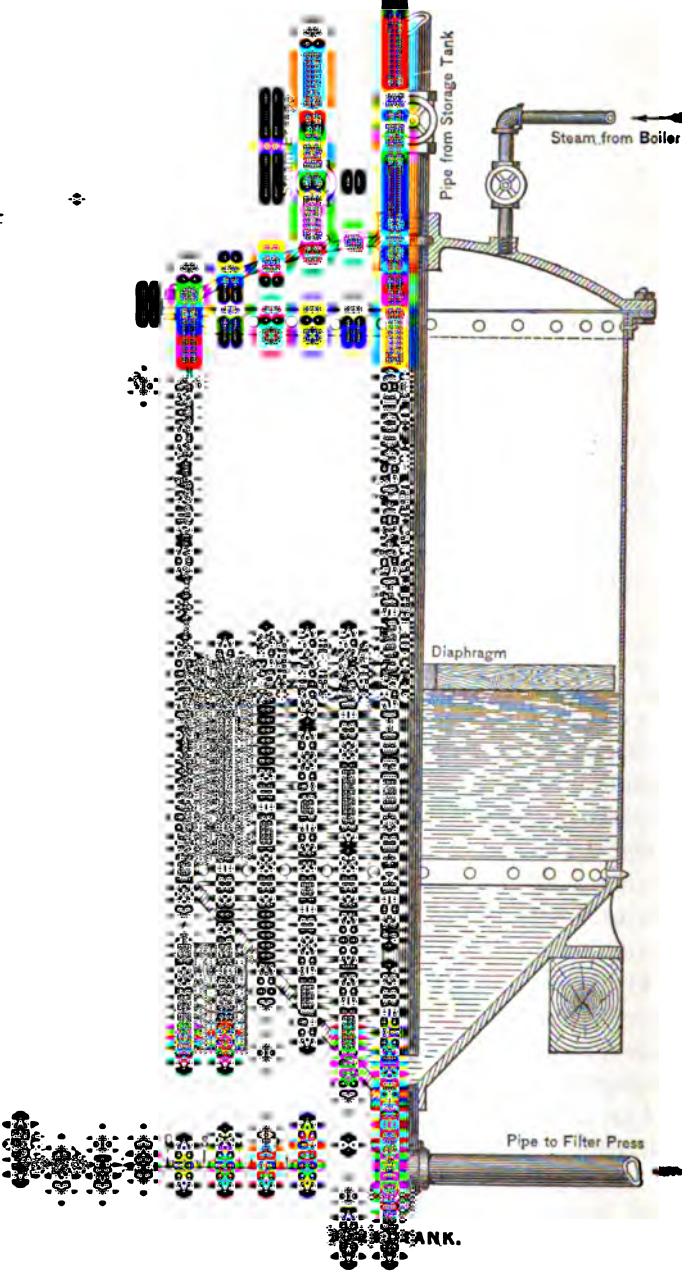
B.—*Leaching- and Storage-Tanks and the Tank for the Preparation of the Extra-Solution.* (Figs. 4 to 17 inclusive.)

All the wooden tanks are usually of white cedar, but this material is not necessary, as was shown at Silver City, the tank lumber being obtained only a few miles from the mill, and put in position while comparatively green. Such a course is not, however, to be recommended. The best plan, except for shipment to Mexico, is to order the tanks made in San Francisco, in diameter about two inches larger than as finally erected, ready to put up, with the exception of having no dowel-pins in the bottom and no gain in the staves. The sides of all the tanks are straight. In making the tanks, even if they have been set up in San Francisco, the staves are dressed to sweep of tank, and of such a length as to allow a 6-inch chime.

The gaining of the staves, one inch deep, is done by hand. No dowel-pins are used in the bottom, but instead the sides are grooved and joined with a tightly-fitting tongue $\frac{5}{8}$ by $1\frac{1}{2}$ inch, which is imbedded in the grooves with white lead.

All joints of the staves and bottoms, and particularly where the bottom enters the gain in the staves, must be fitted with the greatest

PROCESS.


Pipe from Storage Tank
Steam from Boiler
Diaphragm
Pipe to Filter Press

ANK.

accuracy, and the surface forming each joint must be covered with white lead just before finally putting them together. Nails or screws must not be used anywhere. Finally, the tanks are covered outside with three coats of white-lead paint, the last being one of a dark color, such as will not show dirt.

If the lumber is perfectly dry, several coats of paint inside will be beneficial. Too much care cannot be used in the putting together of the leaching-tanks, since not only is an absolutely tight tank required, if the ejectors are to be used to accelerate the leaching, but also because the use of the ejectors will greatly augment any defect in the tank. In the construction of storage- and precipitating-tanks for solution, so much care is not necessary, but will be found advantageous. But the construction of the precipitating-tanks for wash-water requires more care than for any other except the leaching-tank, as the wash-water requires to be heated if the precipitation is by acid and iron. These tanks should be of dry lumber, carefully put together and coated with paint. The dimensions and number of all tanks are given in the drawings, the storage-tanks being of the same dimensions as the precipitating-tanks.

The space under the filter in the leaching-tanks should be as small as possible. The false bottoms for these tanks are prepared as follows: Wooden slats 1½ inches high and 1 inch wide and separated one inch from each other, are fastened to the inside of the bottom of the leaching-tanks by screws imbedded in thick white lead. The side of the slats next to the bottom is cut out, about every 7 or 8 inches, ½ inch deep and 3 inches wide, so that a passage for the solution is established everywhere. Between the end of the slats and the staves a clear space 1⅝ inch wide is left all round, a strip of wood 1½ inches high and 1 inch wide, previously cut by a saw on one side in many places and well soaked in water so that it will bend easily, is fastened around the slats leaving an annular space ⅝ inch wide between the strip and the staves. One thickness of stiff matting (covering the slats and the circular strip, but not the annular space) forms a foundation for the filter cloth proper.

The latter, No. 8 canvas duck, is cut to a diameter 6 inches greater than the inside of the tank so that the edges can be pressed into the annular space described above, and kept in position by forcing down a ½-inch rope. The filter does not require to be removed oftener than once in six or eight months.

There is only one outlet to the leaching-tanks, which is in the center. It consists of a threaded cast-iron flange, bolted to the bot-

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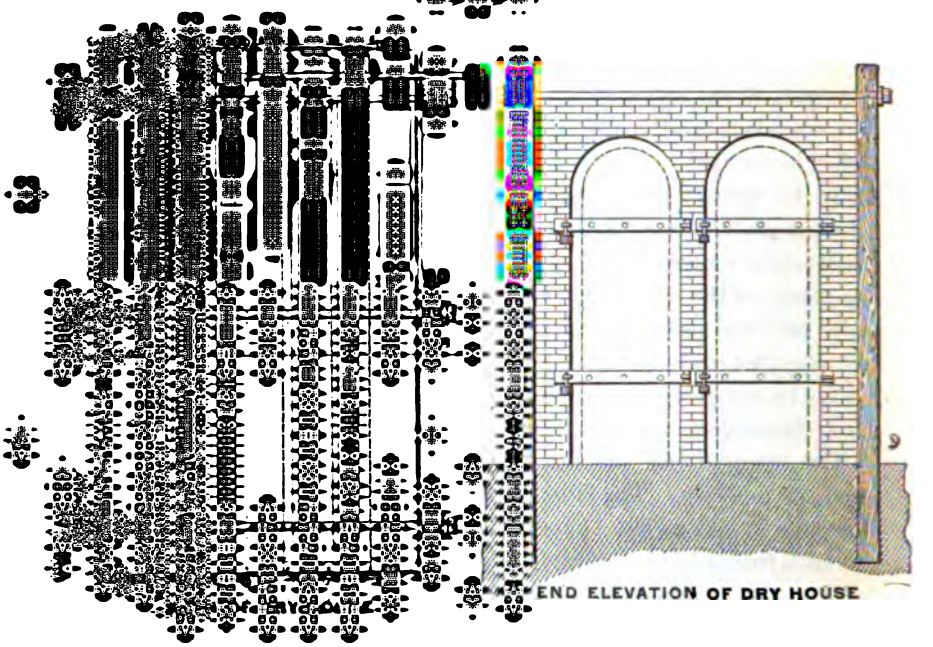
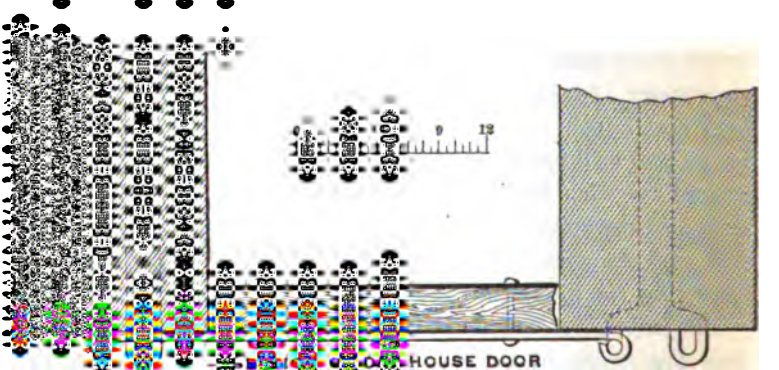
ottom, and together with the
he details of these flanges, and
with them, are given in Figs.



END ELEVATION OF DRY HOUSE



HOUSE DOOR

ses of which vary according to

one of them (H) is always
-solution, and the other two

(G, G) for heating and storing warm solution, which is always used in treating raw ores. Second: If roasted ore is being treated and there is ample leaching-tank capacity, the extra is made up on the ore, the storage-tank (H) being used for heating and storing warm stock-solutions, and the other two (G, G) for cold stock-solution, the coils in the bottom of the latter two not being used. Third: But if the leaching-tanks are to be run to their utmost capacity the storage-tank (H) is used for the preparation of the extra-solution and one of the other two for warm stock-solution and one for cold. The bottoms of the storage-tanks are six feet above the tops of the leaching-tanks. The method of heating the solution is by means of a coil consisting of about 100 feet of 1¼-inch lead pipe in each tank, supported on wooden blocks about 4 or 5 inches above the bottom. In regard to the triple connection between the three storage-tanks, in the first and second cases described above, the valves on the two tanks (G, G) are left open all the time, and the valve on (H) shut all the time.

In the third case all the valves are kept closed.

Above the storage-tanks and just under the ends of the pipe which brings the solution up to the storage-tanks from the solution-pump, are boxes about 2½ feet square and 1 foot high inside, with 1 inch holes in the bottom. The boxes over the two tanks (G, G) are for the purpose of breaking the force of the stream of solution as it comes from the pump, which would otherwise disturb the small sediment of sulphides which is still mechanically held in the solution and is gradually deposited on the bottom of the storage-tanks and is removed once every few months.

This sediment, if disturbed, would pass out of the storage-tanks on to the surface of the ore in the leaching-tanks, and any portion of it deposited after the use of the extra-solution might be thrown out with the tailings. But this box has for tank (H) an additional use if the extra-solution is made up in that tank, as it serves as a chemical box in which the chemicals for the extra-solution are dissolved by the solution passing through it.

At the bottom of each storage-tank around the outlets, are small boxes about 6 inches high, with perforated tops, which act, not only to prevent the solution being drawn down below 6 inches, thus keeping back sediment, but also keep chips, etc., from getting into the pipes and valves, over the leaching-tanks. But if the storage-tank (H) is used for the preparation of the extra-solution, this box must

be removed so as not to keep back any cuprous hyposulphite, or yellow precipitate, which may be on the bottom of the tank.

The hoops of all tanks are of round iron, flattened at one end and with a lug forged on the outer side. The other end of the hoop is upset, threaded, passed through a hole drilled in the lug and a nut screwed on the end. The method of passing the hoop by the sluice-gate is shown in detail in Fig. 12.

C.—*Precipitating- and "Sump"-Tanks and Storage-Tank for Sulphides.*

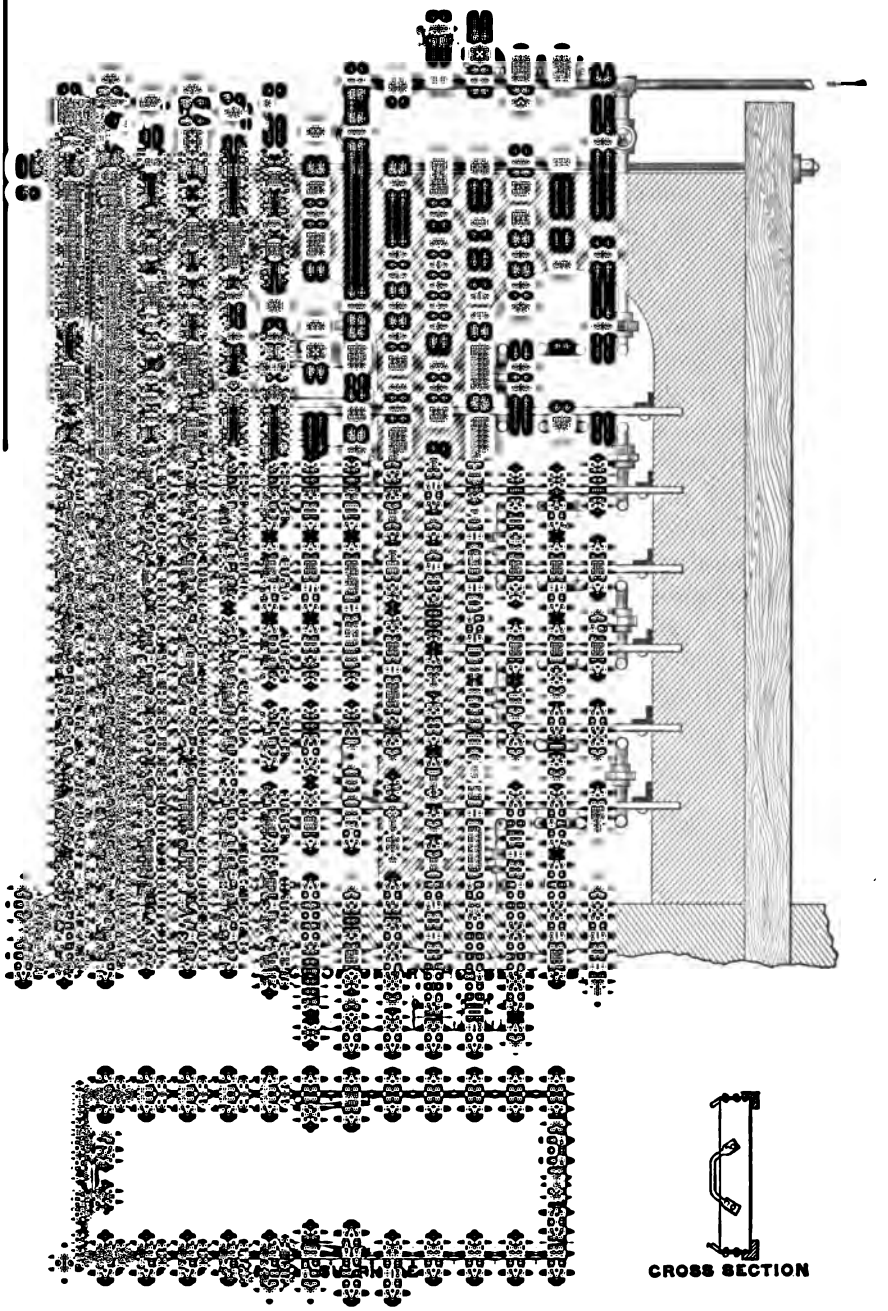
The tops of the precipitation-tanks should be about 4 feet below the inside of the bottom of the leaching-tanks. The number of the precipitating-tanks required will vary according to the following circumstances:

In the treatment of raw ores there is no first wash-water to be precipitated, so that the total number of precipitating-tanks for a raw-ore plant is only three instead of the six shown in the drawings.

In the treatment of roasted ore from which the solution dissolves little or no lead, the number of precipitating-tanks is the same as shown in the drawing.

If lead is to be precipitated three additional tanks will be required, and should be placed between the tanks for the precipitation of solution and those for the precipitation of wash-water. They may be placed in the same line and on the same level, and the solution forced from them into the tanks for the precipitation of silver and copper, by means of siphon-pumps. The details of the surface discharge (d') of the precipitating-tanks are given in detail, Figs. 6 and 7. As to the stirring apparatus, a mechanical stirrer is entirely unnecessary and in the way.

With an ash oar of 16 feet length, the total amount of stirring in twelve hours amounts altogether to only one hour by one man. For easy stirring with the oar, the platform around the precipitating-tanks should not be more than 6 inches below the top of the precipitating-tanks. The "sump"-tank (D) for the solution-pump has a box and strainer around the outlet, for the same purpose as that described under the head of storage-tanks. The storage-tank (E) for sulphides, has no such box. The pipes from the precipitating-tanks should be coated on the outside with asphaltum varnish; particularly is this necessary for the discharge-pipes from the wash-water precipitating-tanks.



CROSS SECTION

D.—*Tanks for the Preparation and Storage of Sodium Sulphide and Soda-Ash.*

The tank, I, for the preparation of sodium sulphide, is of cast-iron, in two sections, the lower being 1¼ inch thick and the upper ¾ inch. The two sections are bolted together through flanges and the joints are leaded. On to the bottom is poured 1 to 1¼ inches of melted lead. The pipe for live steam reaches to within 1½ inches of this lead plate.

The two tanks J, J, for the storage of sodium sulphide are of ⅜-inch boiler-iron, and in the drawing are represented with flanges on top, which are not necessary.

It will be noticed that there is a partition forming a small room around the tank for the preparation of sodium sulphide. This is necessary because in breaking up a drum of caustic soda, pieces are likely to fly and get into the storage-tanks holding the leaching-solution.

If soda-ash is to be used as a precipitant for lead, the same number and size of tanks should be used as for sodium sulphide, and arranged in the same way, but all are of ⅜-inch boiler-iron.

E.—*Pipes, Valves, Launders (or Troughs), Sluice-Gates and Chemical-Boxes.*

The sizes of all pipes are given in the drawings. The only place where the solution has any material effect on iron pipes, is between the outlet of the leaching-tanks and the precipitating-tanks. Therefore rubber hose is used, which seldom requires renewing. Six-ply steam-hose is recommended, simply because it is more durable and does not kink or sag, or wear out at the ends if continually plugged. The line *u*, leading from the center of the leaching-tank to the triple launder, is in one piece. Wooden plugs are used for stopping the ends. The hose *t*, leading from the center of the leaching-tank to the siphon, is the same size as the other, is attached to the flange below the other hose, and must have a gradual slope, without a sag, down to the siphon-pump, so that any drop of liquid coming from the leaching-tank will run down into the siphon-pump.

The hose leading from the siphon-pump up over the leaching-tank is also of the same size and in one piece, and is used in circulating the extra-solution. The end must be movable from one compartment of the double launder to the other. There is also a short piece of hose (*v*) leading from the siphon-pump to the triple

launder, necessary only when the siphon-pump is used to accelerate the leaching. At other times it is kept plugged.

The ends of both pieces of steam-hose lying over the triple launder can be moved from one compartment to another.

The valve in the steam-pipe which furnishes steam to the siphon-pump should be of Jenkins's pattern, and situated 3 or 4 feet above the siphon so that the leaching-solution can not come in contact with it. As described below, the steam for the siphon-pump must be dry and therefore the main and branch lines of steam-pipe must be covered. The 2½-inch pipes (*a'* and *b'*) from the launders to the precipitating-tanks are also steam-hose.

In order to keep any sand or sediment which might get into the launders, from passing into the precipitating-tanks, nipples or short pieces of 2½-inch pipe are inserted in the outlets of the triple launder and project about one inch above the bottom. The launder (*l*) and the pipe (*z*) are for that part of the wash-water which does not require to be precipitated. The trough (or launder) (*m*), at the back of the precipitating-tanks, conducts the sulphides to the storage-tank for sulphides. It has a grade of about 4 feet in 100. The sulphides are run into it from the precipitating-tanks by a short pipe attached to a flange bolted on to the precipitating-tank just above the bottom, so that all the sulphides can run out.

The valve is a gate-valve, something like a molasses or oil-gate, and is shown in detail in Figs. 8 to 11 inclusive.

The launder (*n*, *o*) in front of the precipitating-tanks is horizontal, and has a division opposite a point between the third and fourth precipitating-tanks, so that one half its length is used for conducting solution to the pump-sump, and the other half to conduct the precipitated wash-water out of the mill.

One side of the double launder above the leaching-tanks has no division, but runs the whole length of the leaching-tanks, having a hole over each, which is closed with a wooden plug. This serves for transferring the special extra from one ore charge or tank to another. The other compartment of the launder has a division between each two leaching-tanks and conducts solution or water to the chemical-box, which has a perforated bottom and serves not only as a receptacle for chemicals, but also to break the force of the stream of solution, so that it does not make a hole in the ore.

The sluice-box or launder conducting the tailings out of the mill should have a slope of 1 inch to the foot. The details of the sluice-gate have been given in *Trans.*, xv., but the scale there given was

tank. In using the siphon-pumps for leaching two precautions are necessary: first, the siphon-pump must be upside down (suction up) so that all solution coming from the leaching-tanks, running down the sloping hose, may fall directly into the steam in the ejector. Otherwise, or if the hose has a sag in it, the siphon will act only intermittently, ejecting first steam, then solution, then steam, etc. Secondly, the steam used must be dry, which can only be secured in this case by covering the steam-pipes. The difference in effect between wet and dry steam for siphon-pumps was illustrated at a mill in Arizona. The steam was conducted about 200 feet in an uncovered 1¼ inch pipe. The result was that the siphon-pump would not lift six inches, and was therefore useless. It was then used near the boiler, and lifted 20 feet, and forced the solution an additional 20 feet.

In addition to one siphon-pump for each leaching-tank, there is also one for returning the clear solution coming from the filter-press to the sump of the solution-pump.

If lead is to be precipitated with soda-ash, one siphon-pump (which should be a No. 6) will also be necessary for each of the three lead precipitating-tanks. A siphon-pump is also sometimes necessary in making up the extra-solution in one of the storage-tanks; for in case the chemicals in the box with perforated bottom should not be dissolved by the time the right volume of solution had run into the storage-tank through that box, some of that solution would have to be forced up and circulated through the chemical-box until the chemicals were dissolved.

The solution-pump should be a Knowles 6½ double plunger, with 8-inch steam cylinder, 5-inch plunger and 12-inch stroke, and a 4-inch suction and discharge. The fire-pump for sluicing should be a Knowles (or corresponding size of other make) single cap pattern with 10-inch steam and 5-inch water-cylinder, 12-inch stroke and 4-inch suction and discharge.

G.—*Apparatus for the Mechanical Treatment of the Sulphides.*

This consists of a pressure-tank, Johnson filter-press, and steam-dryer. The details, the pressure-tank, and steam-dryer, are shown in Figs. 1 to 3 inclusive. The filter-press, "circular pattern," is furnished by John Johnson, Franklin Square and Cherry Street, New York. The size should be 18 inches, with 24 chambers, 12 two-inch distance-rings, one dummy-plate, pressure-gauge, indicating as high as 150 pounds, one dozen extra tension-hooks and screws, 3 sets of

"double-chain" filter-cloths, but no pump, pulleys, shafting or hangers. The cost of the above is about $290 f. o. b., New York. Much trouble has hitherto been encountered in using a pump for forcing sulphide into the filter-press, but the pressure-tank gives no trouble. If the pressure-tank and press are arranged as shown in the drawing, very little steam-pressure is needed, as most of the sulphides run by gravity into the press. It is likely that compressed air will eventually take the place of steam for this purpose.

If the sulphides are to be dried by steam the use of the filter-press cannot be avoided. Near the front of the steam-dryer is a small windlass by which the pans of sulphides are lifted into, or removed from the steam-dryer. At the side of the steam-dryer is the sampling-floor and scales. The sampling-floor is best made of iron plates, the larger the better, closely jointed.

H.—*Miscellaneous.*

Experience has shown that a separate steam-boiler for the leaching-plant is advantageous, even when the regular mill-boilers are available. The steam for the leaching-plant can thus be maintained at a greater pressure than would be advisable in the regular mill-boilers, the filter-press, siphon-pumps and steam-dryer doing much better work with high-pressure steam. Moreover, the leaching-plant is thus rendered independent of the rest of the plant, and can be run when the mill-boilers are shut down.

In the rear of the leaching-tanks is a car-track having a slight inclination, which answers not only for bringing ore from the rest of the mill-plant, but also for conveying chemicals from the chemical-room to the leaching-tanks and chemical-boxes if the extra-solution is made up on the ore. The car for chemicals runs over the scales in the chemical-room, where the blue-stone and hyposulphite are stored. The caustic soda and sulphur are in the upper chemical-room, in which is also a pair of platform scales. In addition to the scales already mentioned, there must be another set for weighing the ore delivered to the leaching-tanks.

If the tailings cannot be sluiced out, owing to scarcity of water, they should be removed by cars running under the leaching-tanks, as shown in the drawing.

The whole building should be well lighted, particularly over the precipitating-tanks, a skylight being placed in the roof over each of these tanks.

The platform around the leaching-tanks should be about 2½ feet,

Fig. 12.

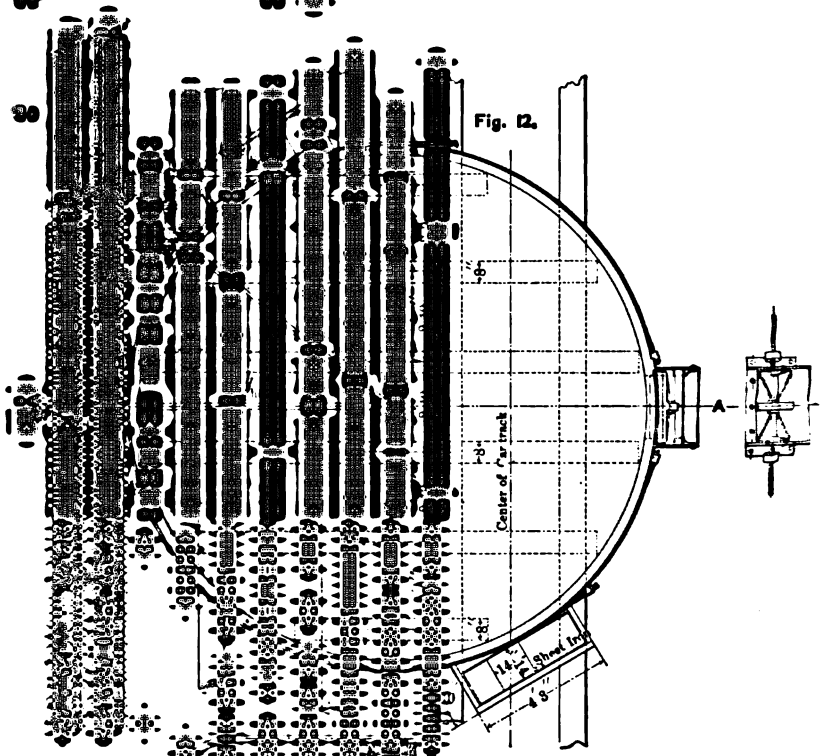


Fig. 13.

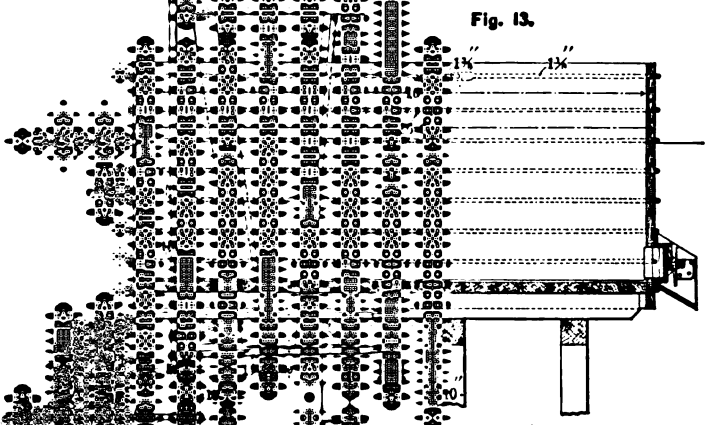


Fig. 12, Plain; Fig. 13, Section.

and that around the precipitating-tanks 4 to 6 inches, below the top of these tanks respectively.

All treatment of the sulphides below the sulphide storage-tank takes place in a room shut off from the rest of the mill.

IV. Expenses.

A.—*Comparative Values of Chemicals in the United States and Mexico.*

The following table gives the average values of chemicals, per hundred pounds avoirdupois, in Mexico, near the railroad, as compared with similar places in the mining regions of the United States.

	Mexico.	United States.
Hyposulphite of soda,	$8.70	$4.25
Blue-stone,	9.50	5.75
Caustic soda,	10.75	5.50
Sulphur,	5.60	3.50
Soda-ash,	7.00	3.50
Sulphuric acid (1700 pound drums),	16.00	3.75
	$9.60	$4.37

The cost of all the chemicals averages about twice as great in Mexico as in the United States. But the use of the acid can nearly always be omitted when the extra-solution is used. The value of chemicals in Mexico therefore averages 8.3 cents per pound, or 1.9 times as much as in the United States.

B.—*Cost of the Stock-Solutions.*

The total amount of hyposulphite for stock-solution for works of various capacities has been given as 3281, 1875 and 1406 pounds for works of 100, 50, and 25 tons respectively.

The corresponding costs in the United States of these stock-solutions are therefore $140, $80 and $60.

C.—*Cost of Precipitating the Silver from the First Wash-Water by Various Methods.*

Table XVII gives the cost of precipitating the first wash-water from roasted ore, and the values of the precipitates. Except in Mexico, where acid costs about five or six times as much as in the United States, there is little difference in cost between the method by acid and iron and that by sodium sulphide. In the United States, if the

wash-water is acid, the cost is in favor of acid and iron by 3 or 4 cents per ton. In cases where the method by dilution can be used, of course the expense of precipitating is merely nominal. But generally, on account of the small cost and greater thoroughness of the first two methods, one of them should be adopted in preference to the method by dilution.

D.—*Cost of Precipitating the Metals from the Leaching-Solutions.*

In all the mills mentioned in Table XXIII, except the Rubio in Parral working on Veta Grande tailings, sodium sulphide was used as a precipitant, and the cost of precipitating per ton of ore, as there given, is the cost of the sodium sulphide.

The comparison in cost between sodium sulphide and calcium sulphide is more favorable for the former in the United States, than in Mexico, as the cost of lime in Mexico and the United States is the same, while the price of caustic soda is very high in Mexico, being nearly—sometimes more than—twice as great as in the United States. Nevertheless, even in Mexico, the sodium sulphide is much superior, both in cost of chemicals and the time, labor and heat required in its preparation. A comparison between sodium sulphide and calcium sulphide, made at Cusi, may be found in the MINING AND ENGINEERING JOURNAL, vol. xlii., page 383, showing the sodium sulphide to be the cheaper by 22 cents per ton. A comparative test at Sombrerete, also made by Mr. Watson, resulted in the adoption of sodium sulphide in place of calcium sulphide, although caustic soda, at that time, cost 14 cents per pound (in Mexican silver).

E.—*Cost of Various Extra-Solutions for Acid Roasted Ores.*

A preceding section gives the weights of blue-stone and hyposulphite required per ton of ore for extra-solutions of various strengths. Table XXIV gives the corresponding costs of these chemicals and extra-solutions per ton of ore. The cost of the extras, including the precipitating of the metals contained in them, varies from 49 cents to $1.24, the latter representing the strongest extra-solution ever used, and used, moreover, under the most unfavorable circumstances, as regards the quantity of hyposulphite required. A corresponding table of extra-solutions for raw ores would show less cost for each extra, because, for the same amount of hyposulphite per ton, the

stock-solution will average in strength about 40 to 50 per cent. higher than in the case of an acid roasted ore, while the volume of extra-solution per ton of ore is less. No table can well be given for simple alkaline roasted ore. A table for alkaline arsenical roasted ores would be about the same as for acid roasted ores.

TABLE XXIV.

COST OF CHEMICALS FOR EXTRA-SOLUTIONS OF VARIOUS STRENGTHS AND OF THE PRECIPITATION OF THE SAME.

	For a Stock-Solution containing $\frac{1}{10}$ of 1 Per cent. Hypo.	For a Stock-Solution containing $\frac{1}{20}$ of 1 Per cent. Hypo.	For a Stock-Solution containing One Per cent. Hypo.	For a Stock-Solution containing 1.2 Per cent. Hypo.	
	1	2	3	4	
Cost of Blue-stone Per Ton	28.7 cts.	28.7 cts.	28.7 cts.	28.7 cts.	For a weak Ex. Solution to contain $\frac{1}{10}$ of 1 per cent. in Bl' stone.
" " Hyposulphite " "	21.2 "	15.9 "	10.6 "	4.2 "	
" " Precipitating " "	16.0 "	16.0 "	16.0 "	16.0 "	
Tot'l Cost of using Ex. Solut'n	$0.66	$0.61	$0.55	$0.49	
	5	6	7	8	
Cost of Blue-stone Per Ton	37.4 cts.	37.4 cts.	37.4 cts.	37.4 cts.	For a medium Ex. Solution to cont'n $\frac{1}{10}$ of 1 per cent. in Blue-stone.
" " Hyposulphite " "	39.3 "	34.0 "	27.6 "	21.2 "	
" " Precipitating " "	24.0 "	24.0 "	24.0 "	24.0 "	
Tot'l Cost of using Ex. Solut'n	$1.01	$0.95	$0.89	$0.83	
	9	10	11	12	
Cost of Blue-stone Per Ton	42.0 cts.	42.0 cts.	42.0 cts.	42.0 cts.	For a str'ng Ex. Solution to contain 1.1 per cent. in Bl' stone.
" " Hyposulphite " "	51.0 "	46.7 "	38.2 "	34.0 "	
" " Precipitating " "	31.0 "	31.0 "	31.0 "	31.0 "	
Tot'l Cost of using Ex. Solut'n	$1.24	$1.20	$1.11	$1.07	

F.—*Comparison Between Raw, Acid-Roasted and Alkaline-Roasted Ores, as to Cost of Chemicals.*

The difference in cost of chemicals per ton of ore is much in favor of alkaline, as compared with acid ores. For the former, the cost of blue-stone is about 27 per cent., of hypo, 43 per cent., and of sodium sulphide, 35 per cent., less than for the latter, as is shown in Tables XIV and XXIII. For raw ores, the average cost of

chemicals is still smaller, being about 20 per cent. less than for alkaline-roasted ore.

G.—*Disposal of the Sulphides.*

The following statistics are furnished by the manager of one of the largest mills using the Russell process, the product being from 60,000 to 80,000 ounces of silver per month.

The monthly shipments are divided into two parts, one being sent to the Chicago and Aurora Smelting and Refining Co. at Chicago, and the other to E. Balbach & Sons, Newark, N. J. The division is made for the double purpose of having two checks on the work of the assayer at the leaching-mill and also of having two refiners as checks upon each other. As the monthly shipments occupy a number of packages, the division is made by sending the even numbered cases to one refiner and the odd to the other, thus securing an average sample of the whole lot for each refiner. The result of this method is that the returns from both refiners are entirely satisfactory, the assays made at the mill and those by the buyers being virtually identical. As soon as the shipments are received by the buyers they are weighed, sampled and assayed, the returns being made within a week after the shipments are received. Within a week after instructions are received by the refiners to work the sulphides, the payments are made by check on New York, the silver being paid for on the basis of New York price on day of settlement, and gold at the rate of $20.60 per ounce.

The Balbach Co. pay 99 per cent. of the value of the silver and full price for gold, and charge $100 per ton for treatment. The Aurora Works pay 98 per cent. of the value of the silver and charge nothing for treatment. Their rates for gold are not known, as the sulphides in question contain only a trace. The Balbachs pay for the least trace of gold. The rate paid for copper is generally $1.00 per unit, although not in the case above mentioned, as very little copper is present.

The sulphides from the Sombrerete mill are treated at the Argo Works, Denver, Col. The price paid by these refiners is 97 per cent. of the gold and silver (silver at N. Y. quotations and gold at $20 per oz.) without any charge for treatment.

The sulphides from the Sombrerete mill contain $700 to $800 in gold per ton. Taking into account the silver entering the cupel, the mill-assays for silver average about 0.6 per cent. above those of the refiners. Without taking this into account, the refiners' assays

are a little higher than the mill assays. The average of the assays on gold agree within 0.1 per cent.

H.—*Details and Summary of Cost of Running the Plant.*

The following statement gives the approximate amount of labor required per day of twenty-four hours, in the leaching department only, for a plant of 100 tons capacity, as demonstrated in various mills:

2 men on leaching-tanks, at	$4.00	$8.00
3 " as helpers, at	2.50	7.50
2 " on precipitating-tanks, at	4.00	8.00
2 " on pump, press, dryer, boiler, etc., . .	4.00	8.00
2 " as helpers, at	2.50	5.00
Total, ...		$36.50
Cost of labor per ton, ...		36½ cents.

The three helpers at the leaching-tanks attend also to the preparation of the sodium sulphide, which occupies about three hours once every two days, and also to the preparation of the extra-solution, which occupies one man about two hours, two or three times each day. Of the three helpers, one is on night-shift, and the other two on day-shift, the sluicing out of the tailings being done only on day-shift, and occupying one man about four to five hours each day. For 50-ton tanks, the two men and three helpers are more than sufficient, as the labor is much less than where ten-ton tanks are used. Also the two men and two helpers are more than sufficient for the solution-pump, press, dryer, boiler, sampling of the sulphides, etc. If the plant is arranged as described, the pressing of the sulphides will occupy only one man three days per week; the removing of the dry sulphides from the steam-dryer and replacing with wet sulphides occupies two men about four hours every three days, and the sampling also two men about six hours every three days. The boiler needs but little attention, the steam required being about as follows: A very small amount nearly all the time for heating the solution in the storage-tank and for the steam-dryer; steam to run two siphon-pumps or ejectors, five hours per day: to press sulphides, three days per week: to run the fire-pump (if tailings are sluiced),

three to four hours each day: to pump the solution to the storage-tanks all day, at intervals; and for the preparation of sodium sulphide one hour every two days. At Yedras, the amount of wood used per day was three-fifths of a cord, for pumping solutions and for making calcium sulphide. Four times this amount (or two and a half cords) would therefore probably be enough for all the above purposes.

As to the removal of tailings, the cost of shovelling out and removal at Cusi, was ten to eleven cents per ton of ore. At Sombrerete, the filling of the tanks from the cooling-floor, and shovelling out and removal of the tailings cost only fifteen cents per ton, labor being cheaper than at Cusi.

At Yedras, the cost of labor and material for repairs on the leaching-plant was only fifty dollars per month or about four and a half to five cents per ton. At Cusi, it was several times as great per ton, owing to the poor arrangement and condition of the plant.

The silver-bearing solution coming from the ore-tanks to the precipitating-tanks should be conducted in rubber hose. But at the Cusi mill this silver-bearing solution was conducted from the ore-tanks to the precipitating-tanks by means of iron pipes, and these pipes, which are practically the only ones attacked by the solution, had to be periodically replaced. The solution-pump was set on top of the solution-sump instead of level with the bottom, and required constant repairing in order to produce a vacuum sufficient to lift the solution. The twenty-two small leaching-tanks and corresponding valves, piping and siphon-pumps also required considerable attention.

For the plant described and for 100 tons per day, the total cost of labor and material for repairs should not exceed $150 to $200 per month, or five to seven cents per ton.

The total cost of the various operations and chemicals required for 100 tons per day is given in the following statements, which show approximately the minimum and maximum running expenses per day of twenty-four hours at the rate of 100 tons per day, in the United States, as shown by actual experience. The cost of charging the ore into the tanks is not included. It would vary probably from ten to eighteen cents per ton. The removal of the tailings is supposed to be by sluicing; if by shovels and cars, the expense may be put at about fifteen cents more per ton.

Approximate Cost of Running Leaching Plant at 100 *Tons Capacity per day of twenty-four hours.*

Minimum.	Per Day.	Per Ton.
Labor, 11 men at $3.00,	$33.00	$0.33
Fuel, 2½ cords at $4.00,	10.00	0.10
Chemicals,	56.00	0.56
Repairs $150 per month,	5.00	0.05
Assaying, ½ total for mill,	7.00	0.07
Total,	$111.00	$1.11

Maximum.	Per Day.	Per Ton.
Labor, 13 men at $3.00,	$ 39.00	$0.39
Fuel, 3½ cords at $6.00,	21.00	0.21
Chemicals,	100.00	1.00
Repairs $450 per month,	15.00	0.15
Assaying, ½ total for mill,	8.00	0.08
Total,	$183.00	$1.83

I.—*Total Mill-Expenses of Various Mills While Using the Russell Process.*

Table XXXVI gives the total mill-expenses per ton of roasted ore in the Cusi mill for nine months, while using the old or ordinary leaching-process, the Russell process, and both processes together (mixed months). The total expenses, including assaying, crushing, drying and stamping, roasting, leaching, roasting and refining of the sulphides, and production of bullion 980 fine, while using the Russell process, were $12.08.

At Parral, also in Mexico, the total expenses for the treatment of tailings from ore which had originally been roasted, and leached by the ordinary process, were $2.10 per ton for 40 tons per day. For roasted ore, at the rate of 10 tons per day, the total expenses were $9.15.

It should be observed that the expenses at Cusi much exceed, are indeed probably double, what they would be in a well-constructed mill in the United States.

In Table XXV the Cusi expenses are taken as a basis, from which are deduced the corresponding expenses for acid-roasted ores and raw ores in the United States.

The Cusi mill, however, was built so as to utilize, in part, old plant, and to work San Bartolo ore by the ordinary process. It was ill-adapted, particularly in roasting and dust-saving facilities, for economical work on the baser San Antonio ore, by any process, and in its leaching department was unsuited to the Russell process. The

$12.08 on which the estimates are based is the average mill-expense per ton while using the Russell process. In the second estimate of Table XXV, the reduction in cost, due to improvements in arrangement and construction, amounts to 30 per cent.

TABLE XXV.

COMPARISON OF TOTAL MILL-EXPENSES PER TON OF ORE IN THE UNITED STATES AND MEXICO, BASED ON ACTUAL EXPENSES AT CUSI.

First Estimate. **Total Mill-Expenses for Cusi Ore and Mill if the same Ore and Mill were in the U. S.**			
Actual total mill-expenses of Cusi (see Table XXXVI)		$12.08	
Deduct for difference in cost of salt and chemicals	$2.54		
Deduct for difference in expense due to shipping instead of refining prod.	.93		
Deduct for difference in cost of supplies, other than salt and chemicals	.60	4.07	
Estimated expense per ton of ore.			$8.01
Second Estimate. **Total Mill-Expenses for Cusi Ore in a Well-Constructed 75-Ton Roasting Mill in the U. S.**			
Total mill-expenses for Cusi ore and mill in the U. S. (see last estimate)		$8 01	
Deduct for difference in cost of running 1 Stetefeldt as compared with 3 Howells	$1.00		
Deduct for difference in expense due to substitution of 6 50-ton for 22 9-ton leaching-tanks	.65		
Deduct for difference in expense due to increase of capacity from 55 tons to 75 tons	.86	2.51	
Estimated expense per ton of ore.			$5.50
Third Estimate. **Total Mill-Expense for Dry-Crushing, Raw-Leaching Mill in the United States.**			
Total mill-expense for Cusi ore in a well-constructed 75-ton mill in the U. S.		$5.50	
Deduct cost of chloridizing, salt and difference in amount of chemicals per ton	$2.40	2.40	
Estimated expense per ton of ore.			$3.10

In a well-constructed raw-leaching mill, the total mill-expenses per ton for 75 tons per day should not exceed $3.00 per ton, and under favorable circumstances would fall as low as $2.50, particularly if the crushing were done by rolls instead of stamps.

At Silver Reef, Utah, the total mill-expense, including also assaying and general expense, was $1.65 per ton for raw tailings, at the rate of 40 tons per day, and $3.00 to $4.00 for raw ore, at the same rate per day. The corresponding expenses for 75 tons per day would be about $1.25 for raw tailings, and about $2.85 for raw ore.

For roasted ore, at Lake Valley, the total mill-expenses for 60 tons per day, were estimated by the general manager, Mr. Hadley, at $4.65 per ton, including assaying, etc.

V. Applicability of the Process.

A.—*Values of the Materials Treated, in Gold, Silver and Lead.*

Table VI gives the average values of all the tailings and raw and roasted ores on which the Russell process has been or is now in use. The values of the tailings, in silver, range from 8 to 12 ounces, of raw ore 14 to 15 ounces, and of roasted ore 12 to 88 ounces per ton. In gold, the value has been from 0 to $5 and $6 per ton. In lead, the range is from 0 to 23 per cent.; ore containing the latter proportion having been treated at Cusi, and also at Sombrerete.

B.—*Special Applicability to Ores Containing Lime, Antimony or Arsenic.*

The superiority of alkaline-roasted, as compared with acid-roasted, ores in regard to amount of chemicals required per ton, has already been illustrated.

So far as extraction by hyposulphite solution is concerned, the most desirable elements which can be present in material which is to be roasted, are antimony or arsenic, or both of them, in combination with the silver; since, in that case, both antimoniate and arsenate of silver are formed during the roasting, both of which are soluble in the leaching-solutions. Of the two elements, arsenic is the most desirable, not only because arsenate of silver is about 3.57 times as soluble by a given weight of hyposulphite as antimoniate of silver, but also because the solubility of arsenate of silver is increased 2.4 times by the presence of caustic alkali, while that of antimoniate of silver is raised only one-sixth. The latter peculiarity causes the presence of arsenic in combination with silver, to be particularly desirable in the treatment of alkaline-roasted ores.

On account of the solubility of antimoniate and arsenate of silver, in the leaching-solutions, some ores can be roasted entirely without salt, and still yield a high percentage by the Russell process (as at Yedras and Ontario).

The Solace ore in Idaho, from which nothing can be extracted in the raw state, yields 92 per cent. after an instantaneous (or 5 seconds) roasting without salt in the muffle, on account of the presence of arsenic and antimony.

C.—*Special Applicability, Either With or Without Concentration, to the Treatment of Raw Ores.*

The advantages in favor of raw leaching as compared with leaching after roasting are as follows:

1st. The cost of plant for raw leaching is much less than that required for a treatment which includes roasting.

2d. Less power and less labor required in handling.

3d. No losses or discrepancies due to dust or volatilization in roasting.

4th. No salt.

5th. No acid, and less chemicals required for leaching.

6th. Less water, the amount per ton averaging only one-fourth to one-fifth that required for roasted ores.

The practicability of raw treatment depends upon the value of the ore, supposing the percentage-extraction to be the same for raw ore of different values. For instance, for a well-arranged mill, the losses occurring in the treatment of raw ore would probably be about 1 per cent., in addition to the loss in tailings, from the time the ore enters the mill up to and including the production of dried sulphides. In a roasting-leaching-mill the corresponding losses would be, probably, about 3 per cent.

In expenses, the advantages are in favor of raw leaching by about $3.00 per ton. If the ore to be treated, therefore, has a value of 20 oz. per ton, 73 per cent. extraction by raw leaching is equivalent to 90 per cent. by roasting and leaching. But if the ore has a value of 50 oz., the extraction from the raw ore must be 82 per cent., to be equivalent to 90 per cent. by roasting and leaching.

The great difference between the result by the ordinary and by the extra-solution, on raw ore, is illustrated in the case of some ores lately experimented upon by Mr. F. M. Watson, Superintendent of the Sombrerete mill. These results were as follows:

Name of Mine.	Value in Ounces.	Per cent. Extracted in Assay Office.	
		By Ordinary.	By Extra.
Buena Vista	60.0	25.0	70.0
San Geranimo	219.0	2.8	81.1
Penoles	38.0	8.0	86.0

If, however, the tailings from leaching, as is frequently the case, contain the precious metals in such a combination as to admit of

concentration, a combined leaching and concentration will frequently be more applicable than roasting and leaching, even though the leaching alone may not yield more than 30 or 40 per cent. of the value of the ore. The additional cost of concentrating the tailings from the leaching will seldom be more than 50 to 70 cents per ton. When leaching and concentration are combined, if the concentration is by the wet method, it must follow the leaching; since, if it preceded, the result would be a separation of the slimes and coarse ore, which would make the leaching practically impossible on account of the mechanical difficulty of introducing the leaching-solutions into or of removing them from the ore.

D.—*Applicability to Tailings from Amalgamation or from the Ordinary Leaching Process.*

The Russell process extracts silver-compounds which cannot be extracted by amalgamation. This is shown in the case of raw tailings at Silver City, New Mexico, which had already been treated twice by amalgamation, and once by concentration, and from which amalgamation, even in assay-office tests, would still extract only 22 per cent., while the extraction by the extra-solution was 72 per cent.

Also at Silver Reef, Utah, the raw tailings, which had already been twice treated by amalgamation and from which practically nothing could be extracted by amalgamation, yield 40 to 80 per cent. by treatment with the extra-solution. This is in spite of the fact, that the Silver Reef ores, from which these tailings were produced, are the only raw ores so far met with, which yield a higher percentage by amalgamation on the first treatment, than by the extra-solution, although the expenses are less for the latter.

At the Minas Prietas, in Mexico, the extraction from the raw ore by amalgamation in a Boss mill, is 76 per cent. of the gold, but only 9.2 per cent. of the silver. The extraction from the tailings by the Russell process is 54.6 per cent. of the gold and 30 per cent. of the silver.

E.—*Applicability of the Process to Dry or Remote Regions, or Where Salt is Expensive.*

The amount of chloride of silver present in an ore is practically immaterial so far as the results of the extra-solution are concerned. This was particularly noticeable at the Ontario, on roasted ore, the mill-extraction by the extra-solution, on ore with a very low chlorina-

tion, being 34 ounces per ton more than could be extracted by the ordinary solution, even in the assay-office (the extraction by the latter representing approximately the amount of silver present as chloride). Also at Yedras, the extra-solution in the mill extracted as high as 28 ounces per ton more than could be extracted by the ordinary solution even in the assay-office; and the difference was much greater, if mill-results were compared. At the Ontario, the mill-results by the extra-solution, on ore roasted with no salt, were only 4 per cent. less than by amalgamation on ore roasted with 18 per cent. salt, and on ore roasted with 9 per cent. salt were 3.5 per cent. higher than by amalgamation on ore roasted with 16 per cent. salt. Hence the Russell process can be used on ore with a very low chlorination, or on ore chloridized with less salt than would be necessary if the ordinary process or amalgamation were to be used. The applicability of the Russell process to remote regions is illustrated by the fact that over 90 per cent. of the leaching-plant, in bulk and weight, is of wood and can be constructed wherever wood can be obtained.

Its great applicability to dry regions, as compared with amalgamation, is shown by the fact that in the treatment of roasted ore leaching requires only one-sixteenth as much water as amalgamation, and in the treatment of raw ores only one-thirtieth as much as amalgamation.

VI. Comparative Applicability of Various Roasting Furnaces to the Preparation of the Ore for the Process.

A.—*Effect of a Very Short Muffle-Roast, With Free Exposure to Air and Heat, on the Extraction and Loss of Silver by Volatilization.*

Table XXVI shows the effect on Yedras ore of a very short roasting in the muffle. In these experiments the ore crushed in the battery through a 26-mesh screen was sifted quickly, or spread in a thin layer on an earthenware dish and exposed in the muffle to a medium red heat. The Table illustrates the fact, that, as far as the extraction by the extra-solution is concerned, the operation of roasting requires not more than five seconds, if each particle of the ore is freely exposed to the heat and furnace-gases at the same time, and that the loss in silver under such circumstances is practically nothing. On the other hand, deeper layers of ore roasted for half an hour to an

hour showed a loss of 11 to 20 per cent., and in the case of concentrates as high as 40 per cent. In the experiments in Table XXVI,

TABLE XXVI.

EXPERIMENTS ON VERY SHORT TIME OF ROASTING. RESULTS OF ASSAY-OFFICE LEACHING-TESTS, YEDRAS ORE.

Thickness of Layer of Ore.	With or Without Stirring.	Per cent. of Salt Used.	Total Time of Roasting.	Value of Roasted Ore. Oz. Silver per Ton.	Per cent. Extracted by Extra-Solution.	Loss of Silver by Volatilizat'n. Per cent.
⅛ inch deep.	Stirring.	7	⅛ minute.	64.3	66.9	0.0
" " "	"	7	1 "	64.0	78.3	0.3
" " "	"	7	1½ "	64.3	75.3	0.0
" " "	"	7	2 "	65.0	.....	0.0
" " "	"	7	2½ "	65.2	77.9	0.0
¼ inch deep.	"	7	½ "	64.5	66.4	0.0
" " "	"	7	1 "	64.6	69.9	0.0
" " "	"	7	1⅛ "	64.6	74.5	0.0
" " "	"	7	2 "	64.8	77.6	0.0
" " "	"	7	2½ "	64.8	86.7	0.0
½ inch deep.	No stirring.	7	15 seconds.	64.5	86.1	0.0
" " "	" "	7	5 "	64.5	83.5	0.0
" " "	" "	3½	5 "	64.5	67.2	0.0
" " "	" "	0	5 "	64.5	56.9	0.0

all the roasted ore was assayed, so that the determination of the percentage lost required no calculation. A still higher extraction in the same time was obtained on Solace ore, but no record of the loss of silver in these experiments was preserved. The above are the only two ores on which the quick roastings have so far been tried.

B.—*Effect of Various Sizes of Crushing on the Furnace-Results.*

Table XXVII compares the Stetefeldt, Howell, reverberatory, and Brückner furnaces as to the results of using various sizes of screen.

The table shows that the Stetefeldt furnace gives about as good results with a 16- as with a 20-mesh screen, and the Howell with a 12- as with a 26-mesh, but the reverberatory gives a little better results with an 8- than with a 16- or a 30-mesh, although the coarse part alone does not give as good results as the fine part alone. With the Brückner, the change from a 26- to a 10-mesh produces very little difference in the results. The table also gives the percentage of salt used in each case. The Ontario ore is of higher grade than the others, and has therefore a little more salt.

C.—*Differences in Extraction Between Ore From Furnaces and Ore From the Cooling-Floor or Vaults.*

The cooling-floor or vault-samples are the only ones which can be taken as a standard of comparison for furnace-work. Evidently

if the percentage of extraction decreases as soon as the ore leaves the furnace, a high extraction from samples taken in the furnace counts

TABLE XXVII.

EFFECT OF DIFFERENT SIZES OF CRUSHING ON THE ROASTING IN VARIOUS FURNACES.

Name of Mine.	Kind of Furnace Used.	Mesh of Screen.	Value of Ore. Oz. Silver Per Ton.	Per cent. of Salt.	Place from which Sample was taken.	Per cent. Extraction by Extra-Solution in Assay Office.
Ontario.	Stetefeldt.	30	75.1	9	Cooling-Floor.	94.6
"	"	20		12½ } 11.7	"	97.0 } 95.2
"	"	16		13½	"	94.1
San Antonio.	Howell.	26	47.0	10 } 10.0	"	89.2 } 89.1
"	"	12		10	"	89.0
San Miguel.	"	26	53.0	8 } 8.0	"	91.9 } 90.5
"	"	12		8	"	89.1
Sombrerete.	Reverberatory.	30	35.9	10 } 10.0	"	87.8
"	"	16		10	"	88.8
"	"	8		10	"	90.5 } 88.5
"	"	8*		10	"	91.8
"	"	8†		10	"	83.6
Yedras.	"	26	54.9	7 7.0	"	85.9 85.9
"	Brückner.	26	61.7	7 } 7.0	"	70.4 } 68.9
"	"	10		7	"	67.5

* Fine half only of the screenings taken.
† Coarse " " " " "

TABLE XXVIII.

FURNACE-SAMPLES FROM BRÜCKNER FURNACE COMPARED WITH THE CORRESPONDING VAULT (OR COOLING-FLOOR) SAMPLES AT YEDRAS MILL.

Final Samples from the Furnace.						Corresponding Samples From the Cooling-Floor.					
No. of Charge.	Mesh of Screen.	Per cent. of Salt.	Per cent. extracted by Ordinary in Assay-Office.	Per cent. extracted by Extra in Assay-Office.	Difference. Per cent.	No. of Charge.	Mesh of Screen.	Per cent. of Salt.	Per cent. extracted by Ordinary in Assay-Office.	Per cent. extracted by Extra in Assay-Office.	Difference. Per cent.
115	10	7	32.5	84.1	51.6	115	10	7	19.0	67.5	48.5
116	26	7	37.7	69.0	31.3	116	26	7	17.9	70.4	52.5
117	26	5	57.0	68.8	11.8	117	26	5	20.3	63.2	42.9
118	26	5	20.0	57.2	37.2						
119	26	7	32.7	63.5	30.8						
120	26	7	40.8	57.0	16.2	119		3	37.9		
121	26	7	53.9	83.8	29.9	to		to		61.8	23.9
122	26	7	76.8 } 50.4	82.4 } 70.4	5.6 } 20.0	126	26	7			
123	26	7	63.1	70.9	7.8	Average.....			32.8	63.2	30.4
124	26	3	54.3	74.4	20.1						
125	26	7	30.0	60.0	30.0						
126	26	7	51.4	71.4	20.0						
Total Average,			45.8	70.2	24.4						
Ave. of 119-126,			50.4	70.4	20.0						

for nothing. Such was the case at Yedras with the Brückner furnace. Table XXVIII, with this particular ore, and the Brückner

furnace, shows that the percentage obtained from cooling-floor samples is 8 to 17 per cent. less than that from the final furnace-sample. On the other hand, the average of a month's run with the reverberatory shows a decrease of only 0.8 per cent. after leaving the furnaces.

The percentage of extraction from ore roasted in a Howell or Stetefeldt increases after the ore leaves the furnace. At Cusi the average increase in extraction from ore roasted in Howell furnaces was 26 per cent after the ore had left the furnace. At the Ontario, with a Stetefeldt, the average was 6 per cent. after leaving the furnace.

D.—*Effect on Brückner Furnace-Results of Introducing the Salt into the Furnace With the Ore, as Compared With Introducing Salt After the Roasting has Begun.*

The experiments in the Brückner furnaces at Yedras may be divided into two classes: first, those in which the salt was introduced into the furnace at the same time with the ore, and, secondly, those in which the salt was added during the roasting.

The first of the above sets of experiments was made particularly to test the applicability of the Brückner furnace to the roasting of the ore for the ordinary leaching process, as that was the process then in use and continued in use for the next six months, on ore roasted in reverberatory furnaces. For this reason, leaching-tests by the extra-solution were not made until afterwards, on vault-samples which had been saved. The average of the assay-office results on these vault-samples was 72.1 per cent. by the ordinary solution and 84.0 per cent. by the extra-solution. This does not include the ore from the dust-chambers, which yielded much less. All the ore produced in these experiments (except the dust) was then leached in the mill by the old or ordinary leaching process, with the results shown in Tables XXIX and XXX. These results in the mill, by the ordinary process, were so extremely poor that the attempt to use the furnaces for the ordinary process had to be abandoned. One great difficulty was the formation of balls of all sizes and of the hardness of brick. Some of these balls were too large to be extracted from the furnace until broken up with crowbars. The percentages of the various sizes of balls were as follows:

Description of the sizes.	Per cent. of each size.
Larger than base-balls,	22.2
Base-balls to marbles,	22.2
Marbles,	9.7
Large gravel,	3.1
Gravel,	42.7
Finer than gravel,	0.1
	100

TABLE XXIX.

THE REVERBERATORY AND BRÜCKNER FURNACES COMPARED BY THE MILL-RESULTS FROM THE OLD OR ORDINARY LEACHING PROCESS, AT THE YEDRAS MILL.

Duration of Mill Run.	In what Furnace the Ore was Roasted.	Value of Ore.	Per cent. by Ordinary in Assay-Office.	Per cent. by Extra in Assay-Office.	Oz. Silver per Ton in Mill-Tailings.	Extraction in Mill Per cent.	Time of First Washing with Water, Hours.	Time of Leaching with Solution. Hours.	Total Time in Hours.	Total Time in Days.
4 Days.	Reverberatory.	50.3 oz.	63.3	76.4	16.6	69.3	17	40	57	2.4
4 "	Brückner.	62.5 "	67.8	79.4	28.4	54.6	11	96	107	4.5
4 "	Reverberatory.	57.1 "	70.6	85.0	14.5	74.6	17	40	57	2.4
4 "	Brückner.	56.9 "	53.0	66.6	44.4	22.2	33	73	106	4.4
4 "	Reverberatory.	63.3 "	72.6	85.3	15.8	75.2	17	40	57	2.4
4 "	Brückner.	69.5 "	66.6	86.8	34.4	50.5	27	128	155	6.4
3 "	Reverberatory.	52.5 "	74.9	86.5	16.6	68.4	17	40	57	2.4
3 "	Brückner.	64.2 "	74.4	91.1	20.5	68.2	24	104	128	5.3
7 "	Reverberatory.	56.7 "	76.6	87.8	12.6	73.6	24	36	60	2.5
7 "	Brückner.	55.3 "	55.1	83.2	25.9	53.2	58	158	216	9.0
6 "	Reverberatory.	60.2 "	74.8	89.8	14.4	76.1	27	46	73	3.0
6 "	Brückner.	67.4 "	79.2	89.0	27.4	59.3	37	98	135	5.6
Aver'ge	Reverberatory.	56.7 oz.	72.1	85.1	15.1	72.8	19.2	40.3	60	2.5
	Brückner.	62.6 "	66.0	82.7	30.2	51.3	32.0	108.0	140	5.9

The second set of experiments, in which the salt was introduced into the furnace after the roasting had begun, showed that the formation of balls could be avoided, but only at the expense of a great decrease in extraction by the ordinary solution, and on considerable decrease in extraction by the extra-solution, even from the furnace-samples, to say nothing of the 8 to 17 per cent. decrease in extraction after leaving the furnace. Tables XXVII, XXVIII, and XXXI give the results obtained in the second set of experiments. Table XXVIII shows the average of the results on vault or cooling-floor samples for the second set of experiments, the average extraction in the assay-office being 32.8 per cent. by ordinary solution and 63.2 per cent. by the extra-solution. The extraction by the ordinary solution being so poor, and the Russell process not having then been introduced, this ore was not leached in the mill. Whether it was subsequently treated, after the Russell process was introduced, or re-roasted in the reverberatory furnace, is not known.

TABLE XXX.

COMPARISON OF THE NET EFFICIENCY OR ECONOMY OF REVERBERATORY AND BRÜCKNER FURNACES WHEN USING THE OLD OR ORDINARY LEACHING PROCESS, AT YEDRAS MILL.

MILL-RESULTS ON ORE FROM REVERBERATORY FURNACES.	
Average Value of Roasted Ore Per Ton (see Table XXIX)	56.7 oz.
Per cent. Extracted in Mill " " "	72.8
Ounces Per Ton Extracted in Mill	41.5
Number of Tons Per Charge	34
Number of Ounces Extracted Per Charge	1411
Average Time of Leaching Per Charge (see Table XXIX)	2.5 days
Number of Charges which could be treated per month	33
Ounces Silver Produced per month	46563
MILL-RESULTS ON ORE FROM BRÜCKNER FURNACES.	
Average Value of Roasted Ore Per Ton (see Table XXIX)	62.6 oz.
Per cent. Extracted in Mill " " "	51.3
Ounces Per Ton Extracted in Mill	32.4
Number of Tons Per Charge	34
Number of Ounces Extracted Per Charge	1102
Average Time of Leaching Per Charge (see Table XXIX)	5.9 days
Number of Charges which could be treated per month	14
Ounces Silver Produced per month	15428

E.—*Comparison of Reverberatory and Brückner Furnaces as to their Economy, and Leaching-Tests on Flue-Dust from Various Furnaces.*

Table XXIX shows the comparison between the results on ore roasted in a reverberatory and that roasted in a Brückner furnace at Yedras, for six periods amounting to 28 days. The table illustrates the difference in assay-office results on the charge-samples, in extraction in the mill by the old or ordinary leaching process, and in the total time required in leaching. The mill-results are in favor of the ore roasted in a reverberatory furnace, to the extent of an additional extraction of 15.1 ounces per ton or 21.5 per cent. of the value of the ore. In the time required in leaching the charges, the difference is in favor of ore roasted in a reverberatory by 3.4 days—the time for reverbe ratory ore being 2.5 days as against 5.9 days for ore roasted in a Brückner furnace. The rate of leaching on ore roasted in reverberatory furnaces and leached without using a vacuum under the filter was three times that of ore roasted in Brückner furnaces and leached with the help of a vacuum.

In Table XXX the above results are arranged in a different form, and illustrate the difference between the two furnaces as to their net efficiency or economy for the Yedras leaching-works, run to their utmost capacity with the ordinary process. The difference in favor of the reverberatory furnaces is about 31,000 ounces per month

additional product, or 203 per cent. more than from ore roasted in Brückner furnaces.

Table XXXa gives statistics of reverberatory roasting at Yedras. The first section given in the table represents the average of 70 days, the second the average of 31 days. Further data of loss of silver in roasting may be found on page 109. Although the furnace losses are not absolutely correct, they are comparatively so.

The method of calculating the loss is so well known that it need not be described here. It is, of course, applicable only in cases where very little dust is formed. Otherwise, the proportion between the amount of ore and dust should be known. The only accurate method is to weigh and assay the ore sent to the furnaces and all the ore and dust obtained from them. Any other method is merely approximate

TABLE XXXa.

RESULTS OBTAINED BY ROASTING IN VARIOUS REVERBERATORY FURNACES AT YEDRAS. AMOUNT OF WOOD USED IN ROASTING, FURNACE-LOSSES AND RESULTS OF LEACHING-TESTS.

Date of Run.	Number of Furnace.	Kind of Furnace.	Cords of Wood per ton.	Per cent. Extracted by Ordinary in Ass'y Office.	Per cent. Extracted by Extra in Ass'y Office.	Per cent. Loss by Volatilization in Roasting.
Section 1, before July 1st, 1887.	1 and 2.	5 Hearth Flat.	.17	76.3	85.8	13.5
	3	5 " Step.	.18	70.9	82.2	17.9
	4	5 " "	.18	72.5	83.1	15.6
	5	3 " Flat.	.25	76.8	86.2	14.8
	6	3 " "	.26	75.5	86.9	15.3
	7	4 " "	.19	78.7	86.9	14.1
	8	4 " "	.20	74.2	84.9	16.0
Section 2, since July 1st, 1887.	1 and 2.			75.5	86.6	7.5
	5			71.8	86.3	7.6
	6			72.7	86.5	8.8
	7			71.2	85.7	4.1
	8			63.4	85.8	3.8

at the best, and is useful only as a means of comparison. In the new mill now being erected by the Sombrerete Mining Co., by the manager of that company, Clarence King, and by W. R. Eckart of San Francisco, particular attention will be paid to the determination of the furnace-losses. The mill will have a screening capacity of 200 tons per day and a leaching capacity by the Russell process of

150 tons. The ore will be crushed with rolls, dried in shelf dry-kilns, and roasted in the largest Stetefeldt furnace so far constructed. All the ore will be weighed and assayed before each operation in the mill, thus affording absolutely correct statistics.

Reverberatory furnaces have so far been used at Sombrerete, and careful statistics have been kept for a year and a half by Mr. Watson, superintendent of the mill. An opportunity will thus be afforded of completing the comparison given in this paper, between the Stetefeldt, Howell, and Brückner furnaces, respectively, and the reverberatory. This is the only kind of comparison (comparing each furnace with the reverberatory) which can probably ever be made, as an opportunity of direct comparison of all four furnaces on the same ore is not likely to occur.

Table XXXb gives the value and the results of assay-office leaching-tests, on samples from the dust-chambers of both the reverberatory and Brückner furnaces at Yedras. The "lower dust-chambers" are the nearest to the furnace. The dust from the reverberatories was much greater in value than that from the Brückner, but in amount was not more than $\frac{1}{10}$ as great. Table XXXc, furnished by Mr. Wilson, superintendent of the Marsaac mill, gives the values, results of assay office leaching-tests, and the comparative amounts of the ore and dust deposited in various parts of a Stetefeldt furnace and dust-chamber, at Park City. By changing the amount of draft, of course, the comparative distribution would be altered. Each of the dust-chambers, except the last, is 10 feet long by 12 feet high. The wing, or arch in the middle of each dust-chamber, reaches half-

TABLE XXXb.

ASSAY-OFFICE RESULTS ON ORE FROM DUST CHAMBERS OF THE REVERBERATORY AND BRÜCKNER FURNACES AT YEDRAS MILL.

	Reverberatory Furnaces.			Brückner Furnaces.			Brückner Furnaces.		
	Salt Added with Ore.			Salt Added with Ore.			Salt Added during the Roasting.		
	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.	Value in Ounces.	Per cent. by Ordinary Solution.	Per cent. by Extra-Solution.
Lower dust-chambers.....	144.2 oz.	46.7	45.8	55.9 oz.	75.1	84.3	72.9 oz.	36.2	66.7
Middle dust-chambers.....	130.5 "	69.1	88.5	57.8 "	63.7	85.3	70.2 "	51.8	76.7
Upper dust-chambers.....	120.2 "	86.0	86.0	114.4 "	60.8	71.8	67.8 "	55.6	80.4

PROCESS.

ances and exits in the diago-
feet square. The comparative
ays' run. In passing through
ing the dust changes its direc-
ght-angle changes of direction
oint, which is nearly always
large ones. For instance, at

Xc.

H A STETEFELDT FURNACE AND
RESPONDING SAMPLES AT
AC MILLS, PARK
.

	5th Dust-Chamber.	6th Dust-Chamber.	7th Dust-Chamber.	8th Dust-Chamber.	9th Dust-Chamber.	10th Dust-Chamber.	11th Dust-Chamber.	12th Dust-Chamber	13th Dust-Chamber.
	1.0 pr ct.	0.8 pr ct.	0.6 pr ct.	0.6 pr ct.	0.6 pr ct.	0.5 pr ct.	0.4 pr ct.	0.4 pr ct.	1.4 pr ct.
	49.7	49.2	52.2	52.6	54.8	50.8	50.8	53.0	51.
	93.3	92.2	93.5	91.8	91.6	91.8	88.6	88.6	86.0

ue averaging about 5 feet wide,
only about four right-angle
a dust-loss at Cusi probably 8
Daly, Lexington, Blue Bird, or
rnace is used. This may be
furnace has ever been erected
, however, are just as necessary

t furnace for roasting has been
ts at the Ontario with this
reen and 17 per cent. salt, and
ay, the extraction on the large
ty of the furnace could not be
e connected with each furnace
uld be crushed. It is no more
st 90 per cent. could have been
of 80-90 tons per day.

The average amount of pulp treated in one Stetefeldt furnace at the Marsaac mill is 70 to 74 tons per day. In this case also, this is the limit of the crushing, and not of the roasting capacity.

F.—*Comparison Between the Howell and the Reverberatory at Cusi.*

The reverberatory furnaces formerly used at Cusi were replaced by Howell furnaces, and the leaching results may be thus compared:

	Mesh of Screen.	Percentage of Salt.	Time of Roasting.	Result by Ordinary Solution in Assay-Office.
Reverberatory.....	30 to 40	5 to 6	18 hours.	80 per cent.
Howell's.....	12 to 20	8 to 10	35 minutes.	85 "

G.—Comparison of the Four Roasting-Furnaces as to the Time Required in Roasting.

Table XXXI shows the difference in time of exposure to the fire required by the different furnaces to produce the given results by

TABLE XXXI.

COMPARISON OF STETEFELDT, HOWELL, REVERBERATORY AND BRÜCKNER FURNACES WITH REGARD TO TIME REQUIRED IN ROASTING AND THE RESULTS OBTAINED.

Name of Mine and Kind of Furnace Used.	By which Process Leached.	Stetefeldt		Howell.		Reverberatory and Brückner.				Per cent. after Lying in Vault or on Floor.
		Per cent. Extraction after 5 Seconds in Furnace.	Per cent. after Lying on Cooling Floor.	Per cent. after 35-45 Minutes in Furnace.	Per cent. after Lying in Vault.	Per cent. after 2-4 Hours in Furnace.	Per cent. after 5-7 Hours in Furnace.	Per cent. after 8-10 Hours in Furnace.	Per cent after 11-15 Hours in Furnace.	
Ontario; Stetefeldt	Russell.	87.5	93.5							
	Ordinary	71.9	87.2							
Cusi; Howell	Russell.			65.6	90.2					
	Ordinary			28.5	78.1					
Sombrerete; Reverberatory	Russell.					29.1	39.4	55.4	90.0	About the same.
	Ordinary					9.4	11.3	14.1	80.9	
Yedras; Reverberatory	Russell.					68.3	64.6	75.1	86.7	⅛ of 1 per cent. less.
	Ordinary					36.5	39.1	58.9	67.9	
Yedras; Brückner	Russell.					71.8	76.4	74.7	80.2	8 to 17 per cent. less.
	Ordinary					49.6	44.3	46.0	36.3	

both extra and ordinary. The times of exposure to the fire, and corresponding final results by the extra-solution are as follows:

Kind of Furnace.	Time of Roasting.	Extraction by Extra-Solution.	
		Furnace Sample.	Cooling Floor.
Stetefeldt	5 seconds.	87.5 per cent.	93.5 per cent.
Howell	35 to 45 minutes.	65.6 "	90.2 "
Reverberatory	11 to 15 hours.	88.3 "	88.3 "
Brückner	11 to 15 hours.	80.2 "	80.2 "

Although higher extraction from furnace-samples can undoubtedly be obtained on some ores, with a Howell furnace in thirty-five to forty-five minutes, or in a Brückner furnace in five to six hours, yet the table shows that, as far as the Russell process is concerned, roasting is, in a Stetefeldt furnace, an operation of only a few seconds—in a Howell, of minutes—but in a reverberatory or Brückner it requires hours.

H.—*General Comparison of the Four Roasting-Furnaces.*

a. Comparison of Reverberatory and Brückner.—The comparative merits of the two furnaces, as illustrated at Yedras, were as follows:

1. *As to the Mechanical Leaching of the Ore.*—In the case of the reverberatory furnaces, whether the salt was added with the ore, or afterward, no hard balls were formed, and the leaching charges remained soft, and were rapidly leached. In the case of the Brückner furnace, the addition of salt with the ore was disastrous, not only converting the ore almost entirely into hard brick-like balls, but producing a hardening of the charge as soon as liquid was turned upon it, so that the mass could not be removed except by picks, thus rendering the leaching extremely slow.

2. *As to Extraction*, the addition of salt with the ore in the reverberatory furnaces produced about the same result as when added in the furnace. But with the Brückner, when salt was added with the ore, the mill-extraction was 17.0 per cent. less than on ore roasted in the reverberatory.

3. *As to Extraction again*, the decrease in extraction from reverberatory ore, after leaving the furnaces, was only 0.8 per cent. no matter when the salt had been added, but the extraction from the Brückner ore, when salt had been added in the furnace, decreased from 8 to 17 per cent. after leaving the furnace.

4. *The Time of Leaching Brückner ore* was 2.3 times as long as the leaching of ore roasted in the reverberatory.

5. *The Mill-extraction from Brückner ore* averaged 21.5 per cent. of the value of the ore less than the extraction from ore roasted in the reverberatory.

6. *The gross product per month* of the Yedras leaching-works, from ore roasted in reverberatories, was at the rate of three times that from ore roasted in the Brückners.

b. Comparison of the Howell and Reverberatory.—The only direct comparison available between these two furnaces was at Cusi, the results of which have already been given. The Howell used 3 to 5 per cent. more salt, but, using a 12- to 20-mesh screen, yielded 5 per cent. higher extraction than the reverberatory with a 30-mesh screen, and in less than one-thirtieth of the time required by the reverberatory. At Cusi was also illustrated, indirectly, the disastrous mechanical effect of the roasting in a Brückner furnace on the leaching of the ore; indirectly, because the one ore, roasted in the Brückner furnace, hardened during the leaching, while, on the other hand, the ore from all the other three producing mines in the camp, roasted in Howell and reverberatories, gave no such trouble, and at the same time yielded a higher percentage of the silver.

c. Comparison of the Stetefeldt with the Other Three Furnaces.—In *a* and *b*, Brückner and Howell furnaces have been compared with reverberatories. The only opportunity of making the same comparison for the Stetefeldt will be at Sombrerete, Zacatecas, Mexico; where reverberatories have thus far been in use, and where a Stetefeldt is now being erected. The results of the very short roasting in the muffle on Yedras ore are, however, a fairly reliable indication that very high results could be obtained in the Stetefeldt with that ore. It should be recollected, also, that it is a peculiarity of the Stetefeldt, as compared with the reverberatory or Brückner, that its effect upon the ore is not limited to the time the ore remains in the furnace, but the extraction from ore roasted in it continues to rise for twelve to fourteen hours after the ore has been taken from the furnace, and at no additional expense. In short, the advantages in favor of the Stetefeldt furnace, as compared with the Howell, reverberatory and Brückner, in regard to the preparation of ore for the Russell process are as follows:

1. *In Capacity.*—The Stetefeldt has a capacity of 70 to 100 tons per day, while that of a Howell rarely reaches 40 tons, a Brückner 20 tons, and a reverberatory 10 tons per day.

2. *In a Better Condition of the Roasted Ore for Rapid and Thorough Leaching.*—As illustrated at Yedras and other places, the effect of a long-continued rolling motion, as in the Brückner, not only often results in the formation of balls, but in a hardening of the leaching-charge, like cement, as soon as water or solution is turned upon it, which was not the case with the reverberatory or the Howell, and never has been with a Stetefeldt furnace.

3. *In Fuel.*—The amount required for roasting in a Stetefeldt is about one-tenth of a cord of wood per ton of ore, while for the Howell it is one-fourth to one-third, and for the reverberatory one-sixth to one-fourth of a cord.

4. *In Power.*—The amount required to run a Stetefeldt is only that required to shake the screen at the top of the shaft, which is probably not more than one-thirtieth of that required to run a Brückner, or one-fifteenth of that required for a Howell furnace, per ton of ore-capacity.

5. *In Extraction.*—Table III shows that the average extraction from ore roasted in the Stetefeldt is 4.2 per cent. above that roasted in the Howell, 4.8 per cent. above the reverberatory, and 6.9 per cent. above that roasted in the Brückner.

6. *In Time of Roasting* (*of Exposure to the Fire*).—Table XXXI shows the time required to obtain approximately the given results.

VII. Comparison of the Russell Process with the Ordinary Leaching Process; with Amalgamation; and with Combined Amalgamation and Concentration.

In discussing the merits of different processes, the rank which any of them shall hold should be based solely on its value *as compared with the others*, not merely on its economic success on any particular material, which is dependent on many conditions entirely outside of the process itself.

A.—*Comparison with the Ordinary Leaching-Process.*

a. At the Ontario Mill, Park City, Utah.—The results of these experiments, Table XXXII, illustrate, not only the greater extraction of silver by the extra-solution, but also the injurious effect of caustic alkali entering the ordinary solution from the precipitant, and show separately the beneficial effects of the use of the acid and of blue-stone, which constitute the most important chemicals peculiar to the

Russell process. In these tests, which were made on a scale of 2 to 3 tons each, the first wash-water being acid, the caustic alkali could not have entered the solution from any other source than the precipitant. The first set of experiments in Table XXXII shows an assay-office extraction by ordinary solution, of 89.0 per cent., but a mill-extraction by ordinary solution of 81.8 per cent. Samples of the mill-tailings were then leached in the assay-office with ordinary solution, which brought the extraction up to only 82.8 per cent. instead of 89 per cent., as it would have been if some of the silver, which had originally been soluble in ordinary solution, had not been rendered insoluble during the leaching in the mill. This insoluble percentage was 6.2 per cent. of the value of the ore.

TABLE XXXII.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT THE ONTARIO MILL, IN 1883-4 (ACID-ROASTED ORE).

Value of Ore. Oz. Silver per Ton.	Per cent. extracted by Ordinary Solution in Assay-Office.	Per cent. Extracted by Extra-Solution in Assay-Office.	Per cent. Extracted in Mill.	Per cent. of Silver Made Insoluble in Ordinary Solution by Caustic Alkali.	Difference Between Extraction in Mill and Extracti'n by Extra-Solution in Assay-Office. Per cent.
AVERAGE RESULTS OF THE FIRST FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
85.0	89.0	90.5	81.8	6.2	8.7
AVERAGE RESULTS OF THE SECOND FOUR MILL-TESTS WITH THE ORDINARY PROCESS. (NEITHER ACID NOR EXTRA-SOLUTION WAS USED.)					
86.5	84.3	92.5	66.2	14.6	26.3
AVERAGE RESULTS OF THE NEXT TEN MILL-TESTS, IN WHICH ACID WAS USED BUT NO EXTRA-SOLUTION.					
90.4	82.6	90.0	82.5	0.0	7.5
AVERAGE RESULTS OF SUBSEQUENT MILL-TESTS, IN WHICH BOTH ACID AND EXTRA-SOLUTION WERE USED.					
72.0	87.0	92.6	92.5	0.0	0.1

During the next four tests, in Table XXXII, the caustic alkali in the stock-solution had increased to such an extent as to cause the mill-extraction to fall to 66.2 per cent., or 18.1 per cent. below the

extraction by ordinary solution in the assay-office. By leaching samples of these tailings in the assay-office, this was raised to 69.7 per cent., which still left 14.6 per cent. discrepancy, the amount of silver rendered insoluble during the leaching in the mill.

Up to this time no part of the Russell process had been used.

The whole stock-solution was now rendered neutral by the addition of sulphuric acid, the amount of acid used to neutralize all caustic alkali which had so far accumulated, being 0.1 per cent. by weight of the stock-solution. Always afterward, the stock-solution was kept neutral by the addition of acid to the extent of from three-fourths to one pound per ton of ore treated. For the next ten tests after the neutralizing of the caustic alkali, and the use of one pound of acid per ton, the extraction in the mill by the ordinary solution varied from the ordinary in the assay-office only 0.1 per cent., showing that caustic alkali no longer had any effect. By comparing these figures with the preceding, it will be seen that the use of an amount of acid costing only three to four cents per ton of ore has raised the mill-extraction 13 to 15 ounces per ton. But the mill-results, although equal to the ordinary in the assay-office, were 7.5 per cent. less than by the extra in the assay-office. The extra-solution was then used in the mill in addition to the acid, with the results shown in the last line of the table. Comparing this with the second line of figures, we see that the Russell process (both acid and extra-solution) made a difference in the mill of 26.2 per cent. which, on 86.5 oz. ore, is 22.6 oz. per ton. As the extra-solution acts as a neutralizer of caustic impurities, less acid is required when the extra-solution is used. On most ores the use of the acid for this purpose can thus be entirely avoided.

TABLE XXXIII.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT SILVER CITY, N. M. (RAW TAILINGS ALREADY TWICE AMALGAMATED AND ONCE CONCENTRATED).

Fineness of the Tailings — Per cent. Passing a Screen of 22,500 Holes per sq. Inch.	Average Rate of Leaching in Inches per Hour with vacuum of 14″ of Mercury.	Value in Ounces Silver per Ton.	Per cent. Extracted by Concentration in Assay-Office.	Per cent. Extracted by Amalgamation in Assay-Office.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Per cent. Extracted by Ordinary in Mill.	Per cent. Extracted by Ordinary and Extra-Solution in Mill.
87.8	0.667	11.6	0	22	42.5	64.7	38	61.7

b. At Silver City, New Mexico (Raw Tailings).—Table XXXIII gives the average results on the raw Bremen tailings at Silver City

for first month's run. The average difference between the results of the ordinary process in the mill and the extra-solution in the mill, was in favor of the extra-solution by 23.7 per cent. of the value of the material treated. The second month, the average difference was 20.9 per cent., and the third month 25.9 per cent. Afterwards, the tailings became too fine for any further leaching, the average rate per hour for the second month being only one-seventh of an inch, and for the third month one-tenth of an inch per hour.

c. *At Lake Valley, New Mexico.*—Table XXXIV gives the average mill-results by the ordinary leaching-process and by the Russell process (two methods). The Table illustrates not only the difference between the results of the ordinary and the Russell process in the mill, but, also, the difference in results caused by using a strong, warm extra-solution with circulation, after the ordinary, as if the ore were an acid ore, as compared with the results of using a cold, weak extra-solution, before the ordinary and without circulation. As shown in the table, the results of the mill-tests were 30 per cent. of the value of the ore in favor of the Russell process, as compared with the ordinary process; although the difference between the ordinary and extra in the assay-office was only 10.5 per cent. The mill-workings by ordinary solution were below the assay-office results by ordinary solution on account of the injurious effect of the caustic

TABLE XXXIV.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS AND WITH AMALGAMATION, AT LAKE VALLEY, NEW MEXICO (ALKALINE-ROASTED ORE).

Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Amalgamat'n in Mill.	Per cent. Extraction by Old Ordinary Process in Mill.	Per cent. Extraction by Russell Process in Mill if Extra was Used after the Ordinary.	Per cent. Extraction by Russell Process in Mill Using a Weak Extra before the Ordinary.
71	81.5	70.6	53	74.4	83

lime in the roasted ore. No separate clean-up was made for the mill-tests by ordinary solution. The clean-up from the Russell process averaged 0.5 to 1.5 per cent. short of that called for by the "apparent" extraction. Although the mill was built for a capacity of 80 tons per day, and the capacity of the leaching part of the plant was shown to be 80 to 100 tons, the crushing part of the plant failed to crush more than 20 tons, thus limiting to that figure the amount which could be treated. The average value of the ore treated for

the first two months was about 17 ounces per ton, and for the last 5 months about 12 ounces. Although the expenses per ton for 20 tons per day were large compared to what they would have been for 80 tons per day, yet this mill continued to run on 12-ounce ore until the supply of ore was exhausted.

d. At Sombrerete, Zacatecas, Mexico.—Table XXXV gives the results of the full course of preliminary mill-tests made at Sombrerete to determine the applicability of the Russell process, the ore being roasted in reverberatory furnaces and leached in charges varying from 325 pounds to 4.5 tons. The first three leaching-tests were made on very fine and pure pyrites alone. On account of its fineness, and not containing any gangue, it became matted during the roasting. Hence the mill-results by the extra-solution fell short of the extra in the assay-office, by 5.9 per cent. The fourth set of mill-tests illustrates the injurious effect, on the results by extra-solution, of wetting down the roasted ore while it is red-hot, the mill-results by the extra-solution falling 4 per cent. below the results in the assay-office by the extra-solution.

TABLE XXXV.

COMPARISON OF THE RUSSELL PROCESS WITH THE OLD OR ORDINARY LEACHING PROCESS AT SOMBRERETE, ZACATECAS, MEXICO (ACID-ROASTED ORE).

Description of Charges.	Weight of Charge.	Value in Ounces Silver per Ton.	Per cent. Extraction by Ordinary Solution in Assay Office.	Per cent. Extraction by Extra-Solution in Assay-Office.	Per cent. Extraction by Russell Process in Mill.
Finest Pyrites from the Jigs, crushed through an 8-screen—3 Charges	325 lbs.	75.8	73.5	90.9	85.0
Coarse and Fine Jig-Products mixed half and half, crushed through an 8-screen—1 Charge	325 lbs.	65.4	67.4	91.3	90.8
Coarse Jig-Products, crushed through an 8-screen—5 Charges	325 lbs.	33.4	76.7	86.2	86.5
Normal Mixture of Jig-Products, crushed through an 8-screen—4 Charges	325 lbs.	44.9	80.6	90.6	91.3
Normal Mixture of Jig-Products, wet down while red hot—3 Charges	4½ tons.	35.4	77.5	84.6	80.6
Normal Mixture of Jig-Products, *not wet down while red hot*—5 Charges	4½ tons.	36.5	74.0	89.3	88.0

At the time of making all the above tests, the mill-results by the ordinary solution were averaging very much below the results by the ordinary solution in the assay-office, so that the actual difference in the mill-results in favor of the Russell process, as compared with the ordinary process, averaged 22.6 per cent. of the value of the ore. The difference in actual clean-up in silver was a little greater, as the

clean-up from the ordinary process fell a little short, while the clean-up from the Russell process was slightly in excess of that called for by the apparent extraction. Since Mr. Watson has had charge of the Sombrerete mill both the mill and the assay-office results have been very much higher.

e. At Cusihuiriachic, Chihuahua, Mexico.—Table XXXVI gives the comparison of mill-results at Cusi by the Russell process and the ordinary leaching-process, during a period of 9 months. The "mixed" months, referred to in the table, are those in which the extra-solution was used on a portion (about one-half) of the charges. During the other months either the extra-solution alone or the ordinary process alone was used for the whole month. The comparison between the two processes is in favor of the Russell process, as follows:

By the use of the extra-solution the time of leaching is reduced 34.8 per cent.; the apparent extraction is increased 6.5 per cent.; the expenses per ton are reduced $1.29; the net mill-profits per ton are

TABLE XXXVI.

THE RUSSELL PROCESS COMPARED WITH THE OLD OR ORDINARY LEACHING PROCESS, AT THE CUSI MILL. COMPARISON OF "ORDINARY," "MIXED" AND "RUSSELL PROCESS" MONTHS FOR NINE MONTHS (SEPT. 12, 1886, TO JUNE 1, 1887).

Months and Process Used.	"Ordinary," "Mixed" or "Russell-" Process Months.	Value of Roasted Ore in Ounces Silver per Ton.	Average Total Leaching Time. Hours.	Per cent. Apparent Extraction in Mill (by Tailings).	Value of Product in Ounces per Ton.	Total Mill-Expenses per Ton, Leached.	Total Mill-Product per Month.	Net Mill-Profit per Ton.	Net Mill-Profit per Month.
Jan. 1887... Ordinary... Feb. 1887... Ordinary...	"Ordinary" Months.	35.1	66	78.3	8.681	$13.37	$19,900	$ 7.81	$ 7,836
Sept. 1886.. ½ R. P..... Oct. 1886... ¾ R. P..... Mar. 1887... ¾ R. P..... May, 1887.. ½ R. P.....	"Mixed" (or part Ordinary and part Russell Process) Months.	36.8	46	82.3	7.985	12.62	27,690	12.42	13,724
Nov. 1886... R. Process Dec. 1886... R. Process Apr. 1887... R. Process	"Russell Process" Months.	39.9	41	84.8	8.201	12.08	42,233	15.11	23,436

nearly doubled; the gross mill-product per month is more than doubled, and the net mill-profits per month are more than trebled. It

will be noticed that the ore during the months when the extra-solution was used was 4 to 5 ounces higher in value than while the ordinary process was in use. Making corrections accordingly, *i. e.*, reducing the value of the ore to 35.1 ounces, the comparison would be in favor of the Russell process by 50 per cent. greater net profits per ton, 86.2 per cent. additional gross product per month and 149 per cent. additional net profits per month.

During the nine months referred to a discrepancy nearly always existed between the apparent and the actual extraction, both while using the ordinary and the Russell process. Part of this (3.2 per cent.) was due to volatilization and loss by dust in roasting the sulphides. This was remedied by substituting steam-drying for roasting. Another portion (2.8 per cent.) was found to be due to incorrect assaying of the sulphides. Another portion was due to incorrect determination of the weight of ore leached. Except in the special tests given in Table XXXVII, the weight of ore actually leached could not be accurately determined, as there were no scales for weighing the ore, and practical difficulties existed in the way of introducing them for general use. The only means of approximately determining the weight of ore treated was by the cubic-foot system, which consists in calculating the weight of the charge from the weight of one cubic foot of the ore from each charge and the number of cubic feet in the charge. Careful experiments showed that this method gave 6 per cent. more ore than was actually present in the charge. But this approximate method was better than none. At intervals careful experiments were made to determine whether any discrepancy really existed between the apparent (*i. e.*, the extraction calculated from the value of the tailings as compared with the value of the ore) and the actual extraction (*i. e.*, the product in silver). In all, six of these tests were made during a period of nine months, the weight of ore used each time, varying from 18 to 94 tons, being carefully determined. All these tests are given in Table XXXVII, of which No. 2 was by the ordinary process. The total amount of ore treated in these special tests was 279.1 tons. The average "actual" extraction (or clean-up) was 87.9 per cent., which is 1.7 per cent. above the "apparent" extraction. No. 2 (of the 6 special tests) shows both an apparent and an actual extraction of over 79 per cent. by the ordinary process.

On San Antonio ore, or on a mixture of San Antonio and San Bartolo, the extraction by the ordinary process averaged much less than this, the difference in extraction between the ordinary and the Russell

TABLE XXXVII.

SIX SPECIAL MILL-RUNS AT CUSI TO DETERMINE THE AMOUNT OF DISCREPANCY BETWEEN "APPARENT" AND "ACTUAL" EXTRACTION IN THE MILL.

No. of Mill-Run.	Kind of Ore Treated.	Mesh of Screen on Battery.	Per cent. of Salt Used.	Total Weight of Ore Leached. Tons.	Value of Ore in Ounces per Ton.	Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra in Assay-Office	Apparent Extraction in Mill. Per Cent.	Actual Extraction in Mill. Per Cent.	Condition of Sulphides.
1	San Bartolo and San Antonio.	16	9	19.6	39.8	85.5	92.0	85.3	84.2	Roasted.
2	San Bartolo and San Antonio.	16	10	18.7	38.4	83.4	91.9	79.7	79.2	Furnace-Dried.
3	San Bartolo and San Antonio.	12	11	46.8	83.9	84.8	91.6	82.6	87.9	Furnace-Dried.
4	San Bartolo and San Antonio.	12	8	94.	41.7	82.7	90.4	85.9	86.5	Steam-Dried.
5	San Bartolo and San Antonio.	26	10	50.	46.9	85.0	89.2	88.5	91.5	Steam- and Furnace-Dried.
6	San Miguel.	26	8	50.	53.0	89.0	91.9	90.4	91.8	Steam- and Furnace-Dried.
	Average.					84.8	90.8	86.2	87.9	

NOTE.—The Second Mill-Run was by the Ordinary Leaching Process.

process being 15 to 34 per cent. of the value of the ore, as shown in Table XXXVIII.

This set of special mill-runs was made to determine the exact difference caused by the use of the Russell process as compared with the ordinary leaching process, both in regard to the value of tailings and in actual product in silver. The ore was mixed—San Antonio and San Bartolo. In order to get the best possible results by the ordinary process, the percentage of salt used in chloridizing was increased, and each charge of ore was leached continuously (some of them for five or six days) by the ordinary process, until for two consecutive days the tailings remained the same and it was evident that no further reduction in the value of the tailings could be accomplished by the use of the ordinary process. The extra-solution was then used on the same charges, and reduced the value of the tailings to the extent of 9.8 ounces per ton. The increase in the percentage extracted, due to the use of the Russell process, was 22.4

per cent. of the value of the ore. The increase in "actual" extraction (product in silver) due to the use of the extra-solution exceeds .

TABLE XXXVIII.

SPECIAL MILL-RUNS BY RUSSELL PROCESS ON CHARGES ALREADY TREATED BY THE ORDINARY OR OLD LEACHING PROCESS, AND ON WHICH THE VALUE OF THE TAILINGS COULD NOT BE FURTHER REDUCED BY THAT PROCESS.

Number of Charges Treated.	Number of Tons Treated.	Value of Ore in Ounces per ton.	Per cent. of Salt Used.	Per cent. Extracted by Ordin'y in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Mill Tailings after Ordin'y Process and before Russell Process.	Mill Tailings after Russell Process.	Mill Per cent. after Ordin'y Process and before Russell Process.	Mill Per cent. after Russell Process.	Difference bet. Ordinary and Russell Process in Mill in oz. per Ton.	Difference in Per cent. of Value of Ore.	Amt. of Silver called for by Difference in Tailings. Total Oz.	Silver obtained in Furnace-Dried Sulphides. Ounces.	Silver obtained in Roasted Sulphides. Ounces.	Per cent. Discrepancy based on Dried Sulphides.	Per cent. Discrepancy based on Roasted Sulphides.
51	450	44.0	11.2	81.2	.....	13.4	6.5	69.6	85.2	6.9	15.6	No separate clean-up in sulphides				
11	97	40.5	10.0	78.1	89.6	15.0	7.2	63.0	83.0	7.8	19.2	"	"	"	"	"
11	82	39.3	12.6	82.5	88.5	15.6	7.8	61.1	81.0	7.8	19.9	640.	747.8	732.	+3.3	+2.9
19	174	48.3	13.5	72.7	86.8	24.5	7.7	49.3	84.1	16.8	34.8	2922.	2890.0	2860.	—1.1	—2.1
92	803	43.0	11.8	78.6	88.3	17.1	7.3	60.7	83.3	9.8	22.4	3562.	3637.8	3592.	+2.2	+0.8

the amount called for by difference in tailings to the extent of 2.2 per cent. in dried sulphides and 0.8 per cent. in roasted sulphides. The difference caused by the use of the extra-solution was 22 to 24 per cent. both in apparent and actual extraction.

f. At Yedras, Sinaloa, Mexico.—Table XXXIX gives the comparison in mill-results of the extra-solution with the ordinary leaching process for a period of two months at Yedras. This was with an experimental run, the Russell process being used on charges of only

TABLE XXXIX.

COMPARISON OF MILL-RESULTS BY RUSSELL PROCESS AND BY THE ORDINARY LEACHING PROCESS IN THE YEDRAS MILL FOR A PERIOD OF TWO MONTHS.

By what Process Treated.	Value in Ounces per Ton.	Per cent. by Ordinary in Assay-Office.	Per cent by Extra in Assay-Office.	Mill Per cent.
Ordinary Leaching Process	60.5	68.9	82.7	64.5
Russell Process.....	63.5	70.6	83.1	82.6

one ton each. The difference in mill-results is 18.1 per cent. or 11.2 ounces per ton in favor of the Russell process. Table XL gives the

comparison between the extra-solution and the ordinary leaching-process as to mill-results, expenses and net profits per ton.

Run No. 1 is the one referred to in Table XXXIX. As no exact record of additional expenses was kept for this run, the expenses are taken the same as in Run 2, in which the record of expenses was carefully kept.

The second run gives the comparison of the ordinary process for five months previous to November 1st, 1887, with the Russell process for the three weeks between November 1st and 22d, the process not having been introduced in the mill until November 1st, 1887. Yedras being so remote and inaccessible, the price of chemicals per pound is very high, the average price being 10.5 cents per pound as compared with 4.75 cents in the mining camps of the United States. The weight of chemicals used per ton is 13 pounds for the ordinary process and 6.1 pounds additional for the Russell

TABLE XL.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS OF THE YEDRAS MILL. COMPARATIVE MILL-RESULTS, EXPENSES AND NET PROFITS.

No. of Mill Run.	Value of Ore. Oz. Silver per Ton.	Ordinary in Mill short of Ordinary in Assay-Office. Oz. per Ton.	Ordinary in Mill short of Extra in Assay-Office. Oz. per Ton.	Extra in Mill above Ordinary in Assay-Office. Oz. per Ton.	Extra in Mill short of Extra in Assay-Office. Oz. per Ton.	Gross Difference per Ton in favor of the Russell Process. Oz.	Additional Cost per Ton of Russell Process.	Net Difference per Ton in favor of Russell Process.
1st.	60-63	2.7	11.3	7.44	0.13	10.10	$1.65	$8.45
2d.	55-64	3.7	9.0	5.33	0.17	8.86	1.65	7.21

process. The average of the runs shows the superiority of the extra-solution to the extent of $9.48 gross profit or $7.83 net profit per ton.

Table XLa gives the comparison for the months of November and December, 1887, the first two months during which it was introduced at that mill. All tests previously made had been on a scale of only one or two tons at a charge. During the above two months the plant for the Russell process was still in process of construction and all the ore crushed and roasted could not therefore be treated by the Russell process, the excess over the capacity of the unfinished plant being treated in the old plant by the ordinary process, thus affording a valuable opportunity of comparing the two

processes. In constructing the new plant very large tanks could not be built owing to the difficulty of obtaining large timber. The leaching-tanks are 11.8 feet diameter and 5.2 feet deep, and hold about 15 tons of roasted ore. The ore was crushed by stamps through a 20-mesh screen, and roasted in reverberatories with 7 per cent. salt. All efforts to roast the ore in Brückner furnaces were entirely unsuccessful, and these furnaces (four of which had been erected at considerable expeuse) have been permanently abandoned. As shown in the tables the mill-extraction at Yedras is not as high as in some other mills. This is due to the fact that the roasting in the reverberatories, owing to the large loss of silver by volatilization in these furnaces, must be conducted with reference to this loss, as well as to the extraction by leaching. The roasting can be so conducted as to yield 90 per cent. to 92 per cent. instead of 83 per cent. of the value of the roasted ore, by leaching; but such a roasting would cause a loss of about 18 per cent. or 20 per cent. by volatilization, instead of 7 per cent.

In another section are given the losses in silver for each of the reverberatory furnaces at Yedras under the management of R. D. Rhodes, the mill superintendent. Considering the long exposure (12 to 16 hours) necessary in roasting in reverberatory furnaces the losses by volatilization are extraordinarily low.

Table XLa gives the comparison of the Russell process and old process at Yedras for November and December, not only as to

TABLE XLa.

COMPARISON OF THE RUSSELL PROCESS WITH THE ORDINARY LEACHING PROCESS, AT YEDRAS, FOR NOVEMBER AND DECEMBER, 1887.

Month.	By which Process Treated.	Value of Roasted Ore per Ton.	Per cent. by Ordinary in Ass'y Office.	Per cent. by Extra in Assay Office.	Per cent. Extraction in Mill.	Weight of Chemicals Used.	Additional Extraction per Ton by Russell Process.	Additional Net Saving per Ton by Russell Process.
November.	Ordinary	56.9 oz.	69.4	82.0	63.7	11.0 lbs.	9.85 oz.	$8.20
	Russell.	55.7 "	70.4	82.6	81.2	20.5 "		
December.	Ordinary	61.15 "	71.9	84.0	67.3	17.4 "	8.89 "	$7.41
	Russell.	58.15 "	67.9	83.8	82.2	21.1 "		

assay-office and mill-extraction, but also as to the amounts of chemicals (calcium sulphide being used for the ordinary, and sodium sulphide for the Russell process) and the additional gross and net saving due to the use of the Russell process, the gross saving being

$9.37 and the net $7.80 per ton. In all clean-ups, so far, the actual extraction in the mill has slightly exceeded the apparent. The value of the sulphides from the Russell process averages 15,000 to 16,000 ounces per ton, with a small amount of gold, the ore containing only a trace of gold. The amount of first wash-water averages 32, of ordinary solution 100, and of extra 14.5 cubic feet per ton.

This amount of extra-solution is unusually large, as the Yedras ore requires more liquid to saturate it than any other.

The rate of leaching is 7 inches per hour, and the strength of stock-solution is maintained at 1.8 per cent. although only one-half pound of hyposulphite per ton is used.

The maintaining of the stock-solution at such a strength with so small a consumption of hyposulphite, is due to the following causes: First, so little base metal occurs in the first wash-water from the Yedras ore, that all the metals are precipitated from it, not only at very slight expense in sodium sulphide, but also with the production of sulphides just as high in grade as those from the regular leaching-solution. Consequently, the amount of silver entering the wash-water is immaterial, and the first weak silver-bearing solution which comes from the tank after the first wash-water, can be allowed to pass into the said wash-water.

Secondly, in determining the point at which the dilute silver-bearing solution shall be turned from the wash-water precipitating-tanks, into those for the regular leaching-solution, Mr. Johnson makes use of the iodine test (the test being made by the man who attends to the precipitating), the weak silver-bearing solution being allowed to run into the wash-water precipitating-tanks until it shows a strength in hyposulphite equal to one-half that of the regular stock-solution. The actual amount of silver extracted per ton of ore by the first wash-water is only 1.2 ounces or 2 per cent.; but the actual amount obtained in the wash-water precipitate, owing to the use of the above method, is 4.5 ounces per ton.

Of course, in using the above method, the first wash-water is not allowed to drain from the charge, but, as it disappears below the surface of the charge, is immediately followed by solution. The use of the above method has reduced the consumption of hyposulphite per ton from 3.6 pounds, formerly used, to about one-half a pound per ton.

The best results on Yedras ore by the extra-solution are obtained by allowing the extra to stand 12 hours in the ore, or by circulating 6 hours and allowing to stand 6 hours.

The extra-solution is made up on the ore, which is charged dry to a depth of 62½ inches, and sinks 8 inches (or 13 per cent.) during the first washing. The depth of extra required is about 24 inches or about 44 per cent. of the depth of the wet charge. When the tanks are charged full, the extra must therefore be made up in 3 charges of 8 inches each, or else made in a separate tank.

The following statement shows comparison of the mill-extraction by the ordinary process at Yedras for 7 months, June to December inclusive, with that of the Russell process for November and December.

MILL-EXTRACTION AT YEDRAS.

Month (1887).	By Old Process.	By Russell Process.
June	71.1 per cent.	
July	66. " "	Process
August	66. " "	not in use till
September	73.1 " "	November.
October	63.1 " "	
November	63.7 " "	81.2 per cent.
December	67.3 " "	82.2 " "

B.—*Comparison with Amalgamation.*

a. At Lake Valley, New Mexico.—Table XXXIV gives the mill-extraction by the Russell process and by amalgamation on roasted ore at the Lake Valley mill, the difference in favor of the Russell process being 12.4 per cent. of the value of the ore. The leaching-charges are 18 to 20 tons, while the amalgamation-charges are 1.5 to 2 tons. The amount of water for amalgamation was about eight times that used in the leaching. The difference in expenses could not be ascertained, as the amalgamation was at the rate of only 3 or 4 tons per day. Probably the difference in expenses would not have been less than $5 per ton in favor of the Russell process. In this table is illustrated a fact which will be again noted, namely, the percentage obtained by amalgamation in the mill agrees very closely with the percentage obtained by the ordinary solution in the assay-office, but bears no relation to the percentage extracted by the extra in the assay-office, while the percentage obtained in the mill by the extra-solution agrees with the latter very closely.

b. At the Ontario in 1883 *and* 1884.—Table XLI illustrates the comparison in mill-results between the Russell process and amalgamation on Ontario roasted ore during Mr. Russell's experiments at

that mill. The amalgamation-tests were necessarily confined to ore crushed through a 30-mesh screen and roasted with 16 to 18 per cent. of salt. But the leaching-tests were made on ore crushed as

TABLE XLI.

THE RUSSELL PROCESS COMPARED WITH AMALGAMATION. RESULTS ON THE LARGE SCALE AT THE ONTARIO MILL, PARK CITY, UTAH, 1883 AND 1884.

Process Used.	Per cent. of Salt.	Size of Screen.	Weight of Charges. Tons.	Per cent. Extracted.
Russell Process	0.	30	2	84.9
Amalgamation	18.2	30	1¼	89.4
Russell Process	0.	30	2	85.7
Amalgamation	18.2	30	1¼	89.3
Russell Process	9.	30	2	96.2
Amalgamation	15.9	30	1¼	92.5
Russell Process	12.5	20	2	97.0
Amalgamation	15.9	30	1¼	92.5
Russell Process	8.	16	2	92.0
Amalgamation	17.9	30	1¼	96.1
Russell Process	12.	16	2	97.1
Amalgamation	17.3	30	1¼	92.5
Russell Process	16.	16	2	95.7
Amalgamation	17.5	30	1¼	95.5
Russell Process	18.	16	2	95.1
Amalgamation	17.5	30	1¼	93.3

coarse as a 16-mesh and with all percentages of salt, from 18 down to 0. The table shows that the results of leaching on ore roasted without any salt were only 4 per cent. less than the results by amalgamation on ore roasted with 18 per cent. of salt. Also, that on ore crushed through a 20-mesh screen and roasted with 12.5 per cent. of salt, the extraction by leaching was 4.5 per cent. greater than by amalgamation with finer crushing and 3.5 per cent. more salt. The average results by leaching on ore crushed through a 16-mesh screen with 12 per cent. of salt, were also higher than those by amalgamation on ore crushed through a 30-mesh screen and roasted with 17.5 per cent. of salt. The amount of water required for amalgamation was 2.3 times as great as that used in leaching. The milling expenses by amalgamation for the year during which these experiments were made averaged about $15 per ton. On the other hand,

as shown by the expenses for a similar ore in the United States, deduced from the Cusi expenses, the corresponding cost in a mill using the Russell process would probably be at least about $5 per ton less.

c. At the Marsaac Mill (Daly Ore), Park City, Utah.—Table XLII gives all the mill-tests so far made in the series of experiments now being conducted by Mr. W. A. Wilson, Superintendent of the Marsaac mill, to test the applicability of the Russell process to the ore treated in that mill. In all cases the ore was roasted for amalgamation—not for leaching. If the ore had been roasted with reference to the Russell process, the results by that process would probably have

TABLE XLII.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE MARSAAC MILL, PARK CITY, UTAH, IN 1887, ON "DALY" ORE ROASTED FOR AMALGAMATION.

No. of Mill-Run.	Total Time Covered by Mill-Runs.	Value of Roasted Ore. Oz. Silver per Ton.	Size of Screen Used.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	October 20th to November 20th, 1887. These tests are now being continued.	33.22	20	10	93.4	94.4	90.0	88.9
2		35.43	"	10	87.2	91.0	90.7	88.5
3		43.03	"	10	89.9	91.2	89.9	87.5
4		40.04	"	10	88.8	90.8	89.0	86.6
5		38.87	"	10	89.4	91.6	90.7	88.2
6		39.34	"	10	86.9	88.9	90.6	86.3
7		40.87	"	10	88.9	90.9	90.5	90.8
8		42.32	"	10	87.1	91.0	93.5	90.8
9		35.94	"	9	88.5	90.7	92.3	90.9
10		40.13	"	10	88.0	90.5	89.8	91.8
11		39.67	"	12	89.4	91.6	89.9	91.0
Averages.					88.9	91.1	90.6	89.2

been 2 to 5 per cent. higher. The weight of ore used in the leaching-charges was 2 tons, and in those for amalgamation 1.5 tons each. The amount of water used in amalgamation is 7 to 8 times the amount used in leaching. The comparison in expenses has not been made; but the Marsaac mill-expenses are about $11, and the difference will probably be $4 to $6 (see table) in favor of the Russell

process, the ore being a simple alkaline ore and requiring only a small amount of chemicals. Experiments are now being made with less salt and coarser screens for the leaching-tests, a 20-mesh screen and 10 per cent. of salt being the least which can be used for ore to be treated by amalgamation.

d. At the Ontario in 1887–88.—Table XLIII gives all the mill-tests so far made in the series of experiments now being made similar to those in the Marsaac mill on Daly ore. These tests also are being conducted by Mr. Wilson to determine the applicability of the Russell process to the Ontario ore. In all the tests, the crushing, the percentage of salt used, and the roasting are fixed with reference to obtaining the best possible results by amalgamation, and not at all with reference to the leaching. Nevertheless, the mill-extraction by the Russell process averages 9.4 per cent. of the value of the ore above the results of amalgamation. The weight of each leaching-

TABLE XLIII.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE ONTARIO MILL, PARK CITY, UTAH, IN 1887, ON ONTARIO ORE ROASTED FOR AMALGAMATION.

No. of Mill-Run.	Total Time Covered by Mill-Runs.	Value of Roasted Ore, Oz. Silver per Ton.	Size of Screen Used.	Per cent. of Salt Used.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assay-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
1	November 20th, 1887, to January 1, 1888. These tests are now being continued.	55.78	26	11	91.6	91.8	73.6	91.8
2		40.28	"	11	83.8	91.9	71.3	92.1
3		36.49	"	13	91.9	94.1	75.5	86.3
4		43.72	"	14	93.1	94.2	86.3	89.5
5		37.06	"	13	93.8	94.4	89.6	94.9
6		39.08	"	12	87.1	91.8	81.1	90.8
7		43.50	"	12	89.5	92.2	89.6	91.3
8		51.40	"	14	83.9	89.9	86.8	89.2
9		43.10	"	15	79.0	91.4	76.6	90.6
10		38.10	"	16	94.0	95.9	81.8	95.1
11		52.90	"	15	93.8	94.2	91.3	94.8
Averages.....		43.76			89.2	92.9	82.1	91.5

charge was 2 tons and that of each amalgamation-charge about 1.5 tons. The Ontario ore has always required about three times as

much water for leaching as any other roasted ore, but still the amount of water used in amalgamation for Ontario ore is 2.7 times that required for leaching the same ore. On all other roasted ores, the amount of water required for amalgamation averages sixteen times that used in leaching. The Ontario mill expenses are about $13 per ton. Those for leaching would be about $5 to $6 less, or a difference in favor of the Russell process of $5 to $6 in expenses, which together with the additional extraction of 4.11 ounces makes a total net difference of $9 to $10 per ton.

For January, as in December, the experimental plant at the Ontario mill was run by Mr. Wilson on Ontario ore alone, it not being considered advisable to run the plant on both Ontario and Daly ore at the same time. The following statement gives the average results for January, 1888, by the Russell process and by amalgamation:

Extracted by:	Per cent.
Ordinary in assay-office,	92.1
Extra-solution in assay-office,	93.8
Russell process in mill,	93.0
Amalgamation in mill,	83.1

The following statement shows the comparison between the extraction by the ordinary solution in the assay-office, and that by amalgamation in the mill, for the last ten days in January.

Ordinary solution in assay-office,	92.0
Amalgamation in mill,	78.1

The best results by the Russell process on Ontario ore were obtained while using a stock-solution of about 1.5 per cent. hyposulphite, at a temperature of 90° to 120° F., and with an extra-solution of about the same temperature, and using about 6½ pounds of blue-stone per ton. The weight of charge treated was 2 tons. The depth of the dry charge before leaching was 24 inches and after leaching 18 inches. For 2 tons of Daly ore, the depth is 22 inches and after leaching 16 inches. The temperatures of the solutions are the same as for Ontario ore; but the amount of blue-stone used is only 4½ pounds per ton. Although both Ontario and Daly ore yield an alkaline first wash-water, the extra-solution can be circulated (and usually is), as if they yielded an acid first wash-water. The amount of gold extracted by the Russell process from both Ontario and Daly ores is the same as by amalgamation. The amount of gold in the ore

is, however, quite small, amounting to only about 0.1 ounce per ton for shipping-ore and probably about one-half that amount for milling-ore.

e. At Cusihuiriachic, Chihuahua, Mexico.—Table XLIV gives the comparison in mill-results, both in apparent and actual extraction, between the Russell process and amalgamation. The ore for each test was weighed with the utmost care. In this case also, the crushing, use of salt, and the roasting were conducted solely with reference to obtaining the best results by amalgamation. The results showed, in the case of amalgamation, a discrepancy between the apparent and the actual extraction, the actual extractions being 6.7 per cent. less than the apparent. The results of leaching, on the other hand, showed a *plus* discrepancy, the actual extraction averaging 2.2 per cent. more than the apparent. The comparison in actual extraction in silver is in favor of the Russell process by 11.4 per cent. of the value of the ore or 5.7 ounces per ton. The difference in expenses between the Russell process and amalgamation is less for Cusi than for most other ores, being about $2.50 per ton in favor of the leaching, making a total net difference of over $8 per ton in favor of the Russell process.

TABLE XLIV.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AT THE CUSI MILL. SPECIAL MILL-RUNS ON SAN ANTONIO AND SAN MIGUEL ORES.

Name of Ore Treated.	Mesh of Screen Used.	Per cent. of Salt.	Value of Ore. Oz. Silver per Ton.	Per cent. of Salts Soluble in Water.	Per cent. by Ordinary in Assay-Office.	Per cent. by Extra in Assay-Office.	Number of Tons Treated.	Apparent Extraction in Mill. Per cent.	Actual Extraction in Mill. Per cent.	By which Process.
San Antonio and San Bartolo	26	10.	46.96	16.5	85.0	89.2	50	88.5	91.5	Russell Process.
							50	86.3	79.6	Amalgamation.
San Miguel.....	26	8.	58.08	9.5	89.0	91.9	50	90.4	91.8	Russell Process.
							50	87.5	80.8	Amalgamation.

C.—*Comparison Between the Russell Process and Combined Amalgamation and Concentration.*

All the foregoing comparisons have been based on mill-results. On the ore referred to in Table XLV, the data for which were furnished

by Mr. G. J. Rockwell, no tests have yet been made on the large scale by the extra-solution, hence the comparison is between mill-results by amalgamation and concentration, on the one hand, and assay-office results by the Russell process on the other. The previous comparisons have been on roasted ore, but this is on raw ore. As no difficulty has been experienced elsewhere in obtaining as good results (within 2 to 3 per cent.) in the mill as in the assay-office on raw ore by the extra-solution, there is no reason to expect any in this case. Table XLV shows in Run I a difference in favor of the Russell process of 53.5 per cent. of the value of the ore as compared with the Boss process, and of 34.8 per cent. of the value of the ore as compared with the combined Boss process and concentration. In Run II no leaching-tests by extra-solution were made. In Run III, the difference is in favor of the leaching as compared with the Boss process by 39.6 per cent. of the value of the ore, and 38.3 per cent. as compared with the combined Boss process and concentration.

TABLE XLV.

COMPARISON OF THE RUSSELL PROCESS WITH AMALGAMATION AND WITH COMBINED AMALGAMATION AND CONCENTRATION, AT THE BIG SANDY MILL, ARIZONA.

Number of Run.	Description of Run.	Value Oz. Silver. per Ton.	Per cent. Extraction by Concentration.	Per cent. Extraction by Amalgamation.	Total Mill-Extraction.	Per cent. Extraction by Ordinary in Assay-Office.	Per cent. Extraction by Extra in Assay-Office.
I.	4 Days Run without Chemicals.	21.8	28.7	25.3	54.0 pr ct.	49.0	85.1
	3 Days Run with Chemicals..... Amalgamation by Boss Process *before* Concentration.	21.5	8.7	45.3	54.0 "	50.5	92.5
II.	4 Days Run ..... Concentration *alone.*	20.8	43.0		43.0 "	No leaching tests made.	
III.	6 Days Run with Chemicals..... Amalgamation by Boss Process *before* Concentration. Coarse Ore Concentrated, Fine Ore Amalgamated.	30.2	63.7	51.9	53.2 "	50.4	91.5
IV.	3 Days Run with Chemicals..... Amalgamation by Boss Process *after* Concentration.	25.0	44.8	7.6	52.4 "		87.8
V.	4 Days Run with Chemicals ..... Amalgamation by *Chlorge* Process after Concentration.	20.7	34.4	28.8	63.2 "	No leaching tests made.	

Run IV shows in favor of the Russell process 43 per cent. when compared with concentration, or 35.4 per cent. compared with concentration and Boss process combined.

The average difference in assay-office results by the ordinary and extra-solutions is in favor of the extra-solution by 38 per cent. of the value of the ore. Here also, as has been noted in the case of roasted ore, the amalgamation-results agree closely with the results by the ordinary solution.

D.—*Relation of the Mill-Results Obtained by Amalgamation, and Those by the Russell Process, to the Results Obtained in the Assay-Office by the Ordinary and Extra-Solutions.*

In making the comparisons between the mill-results by amalgamation and by the Russell process, the relations of the former to the ordinary and of the latter to the extra in the assay-office, have been noted. Table XLVI gives the results of mill-runs on five different kinds of ore by amalgamation and by the Russell process, and the corresponding assay-office results by the ordinary and extra. The extractions from San Antonio and San Miguel are actual clean-ups in the mill, the others being apparent extractions in the mill.

TABLE XLVI.

RELATION OF MILL-RESULTS BY AMALGAMATION AND RUSSELL PROCESS TO THE ASSAY-OFFICE RESULTS BY THE ORDINARY AND EXTRA-SOLUTIONS.

Name of Ore.	Per cent. Extracted by Ordinary in Assay-Office.	Per cent. Extracted by Extra in Assy-Office.	Per cent. Extracted by Amalgamation in Mill.	Per cent. Extracted by Russell Process in Mill.
Lake Valley	71.0	81.5	70.6	83.0
San Antonio and San Bartolo	85.0	89.2	79.6	91.5
San Miguel	89.0	91.9	80.8	91.8
Daly	88.9	91.1	90.6	89.2
Ontario	89.2	92.9	82.1	91.5
Average	84.6	89.3	80.7	89.4

The mill-results by amalgamation average 80.7 per cent., which is 3.9 per cent. below the ordinary in the assay-office. On the other hand, the mill-results by the Russell process average 89.4 per cent. or 0.1 per cent. above the extra-solution in the assay-office, and 8.7

per cent. above amalgamation in the mill. These results indicate that amalgamation-results on roasted ore are almost as dependent on a good chloridizing roast as is the ordinary leaching process. But the Russell process, on the contrary, may be entirely independent of it.

VIII. Summary, Being a General Comparison of the Russell Process With the Ordinary Leaching Process, and With Amalgamation.

A.—*With the Ordinary Leaching Process.*

1. The Russell process permits a coarser crushing of the ore.

2. It requires a less careful chloridizing roasting. Hence less salt, and sometimes no salt, need be used.

3. The roasting may be accomplished in a suitable furnace almost instantly; but the roasting for the ordinary process requires time.

4. The results obtainable by the ordinary process are dependent upon a good chloridizing roast, while the Russell process is sometimes entirely independent of the chloridization, and always exceeds in extraction, the amount actually chlorodized.

5. The extra-solution allows a much more systematic leaching and gives more regular results.

6. The results by the extra-solution are not affected by the presence of caustic alkali in the ore or solution, while even minute quantities affect very powerfully the results by the ordinary process.

7. The Russell process is much more applicable to ores containing lime or lead.

8. It produces sulphides free from lead, and the lead as a by-product.

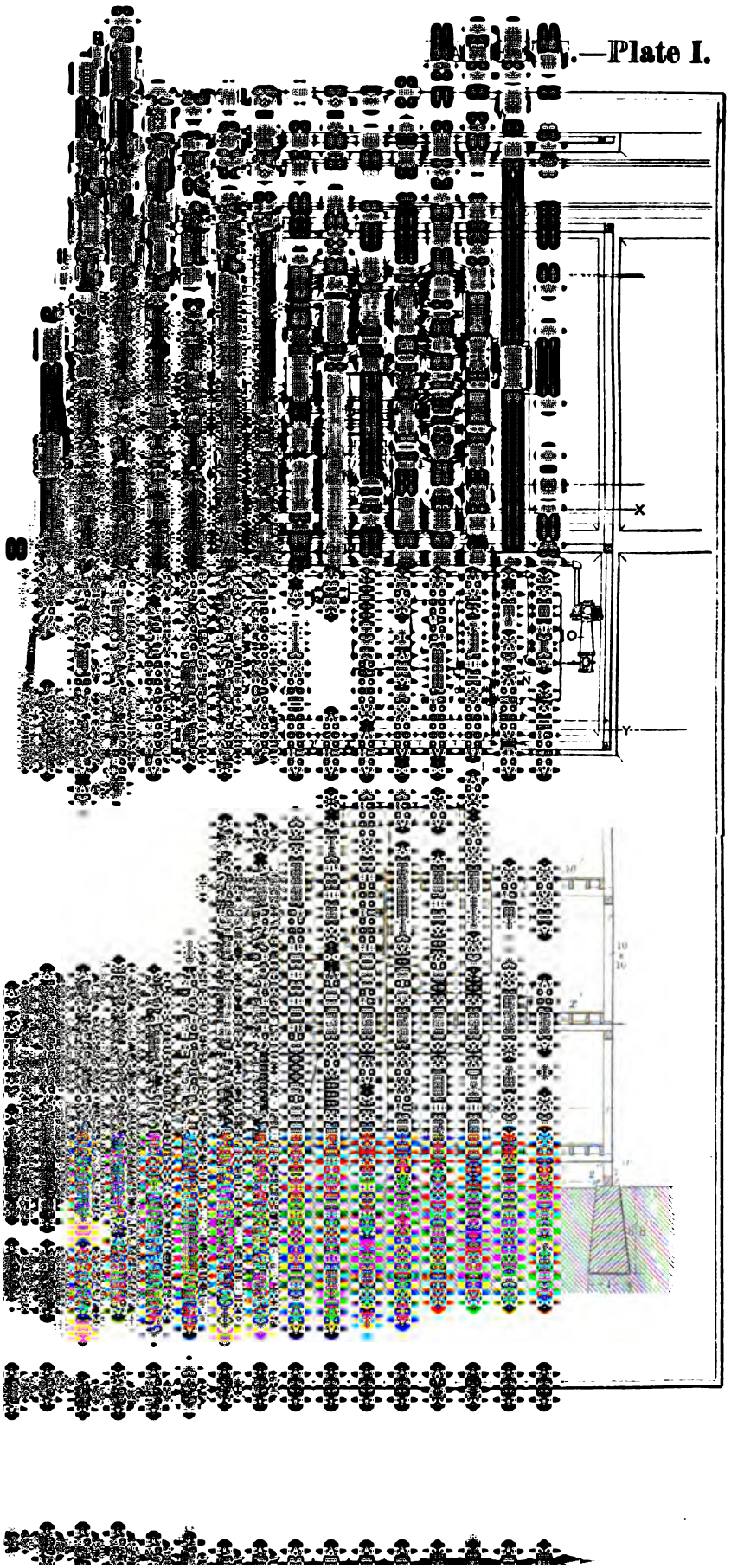
9. From raw ores it extracts often two to ten times as much as can be extracted by the ordinary solution.

10. Hence, ores which would require chloridizing for treatment by the ordinary process can often be treated raw by the Russell process, thus allowing the erection of a cheaper plant and saving the additional expense and losses incident to chloridizing.

B.—*With Amalgamation.*

1. The cost of plant for the Russell process is much less than that for amalgamation.

2. It permits a much coarser crushing of the ore, the size required



[illegible].—Plate I.

(Fig. 4.)

for amalgamation being rarely less than a 26-mesh and usually a 30-mesh screen, while leaching, on the other hand, requires only an 8- to 16-mesh screen.

3. The results of amalgamation are closely dependent upon a good chloridizing roast, while the Russell process is often independent of it.

4. Hence a less careful roasting is required, and therefore less, and sometimes no, salt.

5. The roasting for amalgamation requires time, while that for the extra-solution can be accomplished almost instantly.

6. The extraction in the mill by the Russell process is almost always higher, and the expenses are always less, than by amalgamation.

7. Raw ores that can be successfully treated by amalgamation yield a higher percentage by the extra-solution (except Silver Reef) and always at less expense.

8. The Russell process permits the extraction, from roasted ore, of copper and lead as valuable by-products.

9. It is not injurious to the laborers' health, as is apt to be the case with amalgamation.

10. It extracts compounds of silver which are not attacked by amalgamation.

11. It requires no machinery in the leaching, and no moving parts, except a single pump.

12. The charges treated are from ten to thirty times the size of those treated by amalgamation.

13. The amount of water required for roasted ores averages only one-sixth, and for raw ores only one-thirtieth of that required for amalgamation.

14. In large mills, the quantity of quicksilver in circulation represents a capital of $30,000 to $40,000, which is more than 150 times the cost of the stock-solution for the Russell process.

TWENTY YEARS' PROGRESS IN THE CONCENTRATION OF SULPHURIC ACID.

BY W. H. ADAMS, M.E., NEW YORK CITY.

(Duluth Meeting, July, 1887.)

ONE of the most attractive subjects for technical writers is the gigantic industry of the manufacture of sulphuric acid. This is no doubt, natural when we take into account that it has grown in this country during the past twenty years, from a production of 60,000 tons in the year 1867, to that of nearly 450,000 tons in 1887.

We may well excuse much of the literature which but repeats the same old story, as it serves to bring to the public notice, at proper intervals, the hidden resources of our land; and this constant application of essential truths will sooner or later be the means of freeing us from dependence upon the Old World for necessary chemical products. The present increase in manufacture of this acid in the United States is almost wholly from pyrites ores, which we have in unlimited quantities close at hand. The largest consumption of such acid is for treatment of other native products, petroleum, rock-phosphates, etc.; and, considering the knowledge we have gained during the past twenty years, it is safe to say that the industry is but fairly established, and the causes which have operated to place us in our present proud position are but natural incentives for a further growth and an increased consumption of the many other undiscovered or undeveloped resources of our American soil.

In one of his earlier editions Muspratt says: "It would seem that the manufacture of sulphuric acid, though as extensively carried on as any other business, yet seems to be the least understood; and the ignorance of some manufacturers appears to be only equalled by that of writers, who, in their description of the production of this acid, give figures of apparatus, and vaunt as improvements, things which have been exploded for years. It is possible to find in the works of eminent men drawings of apparatus, for the production of sulphuric acid, which never could possibly have worked one day—some, in fact, could never have worked at all."

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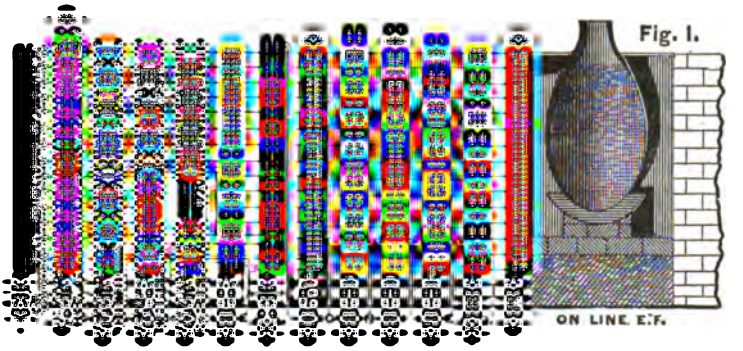
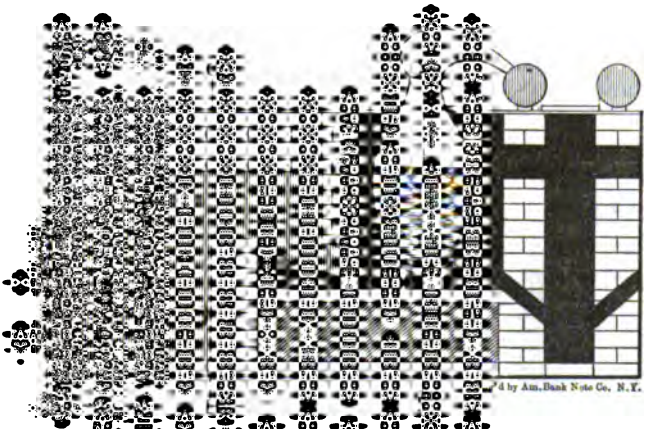


Fig. 1.



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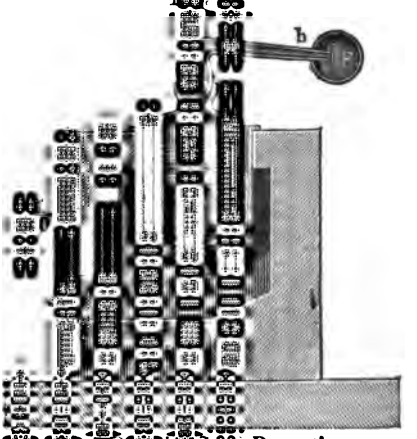

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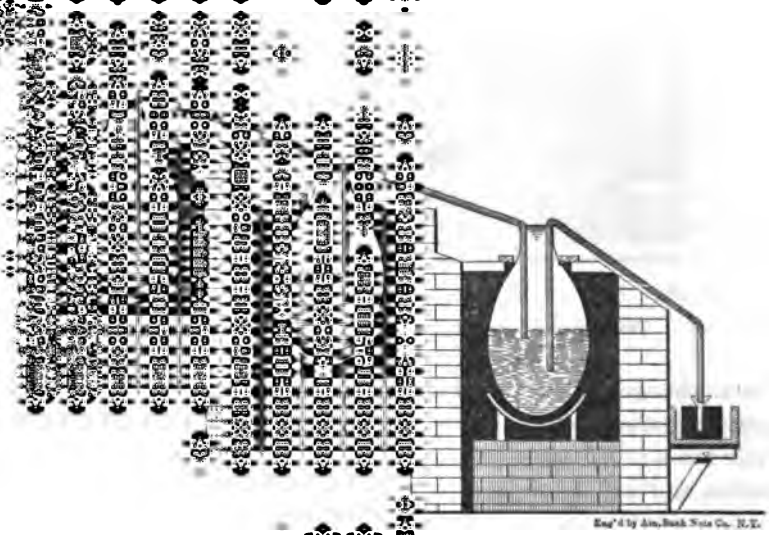

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Continuous Process of 1869

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adjust and harmonize these adjuncts has brought about savings and betterments in all departments. Notwithstanding the great value these towers possess, and the part they have played in lessening the cost of chamber acids, it is yet a fact that less than one-half our American works have adopted them.

In other departments, however, we have much of progress to show for two decades, and many radical improvements of our own to account for the well-understood and significant truth, that sulphuric acid is selling in America at as low prices as are ruling in Europe, though the quantities manufactured are so disproportionate.

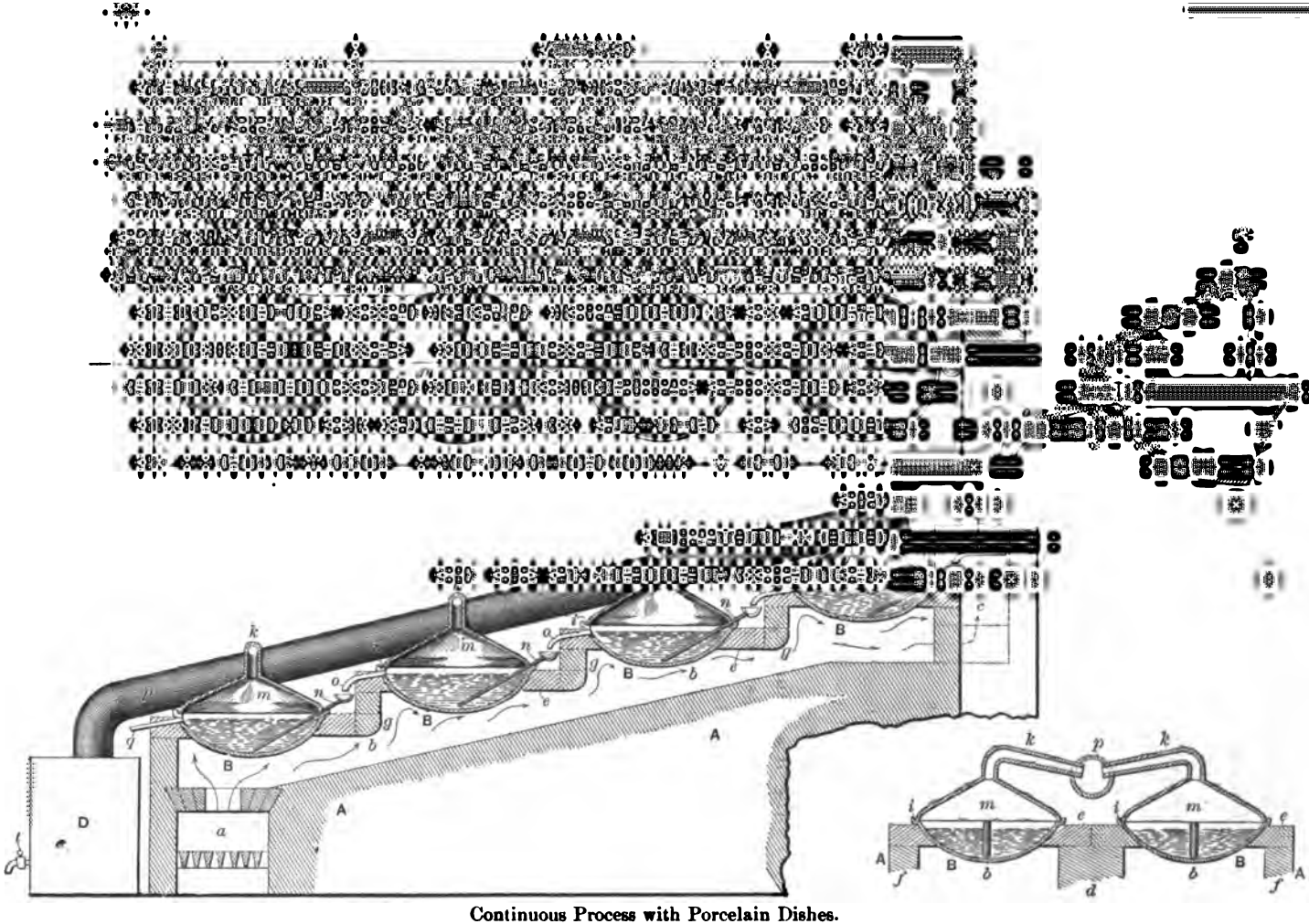
Perhaps this statement will be more fully appreciated if the main raw product, which enters into the manufacture, is brought into comparison, as to sale-prices. Thus:*

A unit of sulphur, in brimstone, in Europe, costs	16 to 22 cents
A unit of sulphur, in brimstone, in America, costs . . .	18 to 24 cents
A unit of sulphur, in pyrites, in America, costs	5 to 12 cents
A unit of sulphur, in pyrites, in Europe, costs	3 to 10 cents

The problem to be solved in concentrating the acids produced in the chambers, say 50° Baumé, is simply the driving off of the excess of water in the form of steam. For this purpose shallow pans, shaped from single sheets of rolled lead, are commonly used. They are of convenient length and width, and the thickness of the sheets is proportioned to the service. Open tops are the rule, and this necessitates firing them from the under side.

Without noting the early details of practice, it will be sufficient to give a general idea of the best construction now used, consisting of a series of six pans, each pan having a rise of two inches above its fellow. Counting from the fires, pans Nos. 1 and 2 are of 18-pound lead set into a bed of cement, levelled from fire-brick arches which span the flues. Nos. 3 and 4 are of 15- and 10-pound lead, and are set upon cast-iron plates which span their respective flues. Nos. 5 and 6 are of 8-pound lead, and rest upon perforated cast-iron plates, which span their flue-spaces. The construction upon which the leaden pans rest gives the greatest safety for weights carried, and at the same time a fair conductivity. Chamber-acids are con-

* Differences in price per unit are owing to variations in the charges for transportation of the ores, etc., from sources of supply, and competition at trade-centres. To this difference in price of sulphur, may be added the well-known differences in labor, fuel, interest, etc.; items of cost with us always greatly in excess of Old World prices.



Continuous Process with Porcelain Dishes.

stantly fed into the upper pan, and, parting with their water, are removed immediately over the fires at a strength of, say, 61° to 62° Baumé.

The advantages of this class of construction are: (*a*) Low cost of the plant. (*b*) Convenience for repairs and regularity of operation by ordinary labor. (*c*) All parts are in plain sight of the manager in his daily rounds, and of the workmen at all times.

The disadvantages are: (*a*) An excess of fuel used for evaporation, the chimney-losses averaging fully 25 per cent. of the coal burned. (*b*) The heats are so irregular, notwithstanding every care in firing, that greater destruction of materials of construction must be allowed for. (*c*) The floating dust and dirt, inseparable from the process at all times, contaminate the acids.

The adoption of gas-firing, which gives uniform heats without necessary waste, and reduces the wear and tear to a minimum, will overcome much of this disadvantage.

The best practice, however, is admitted to be by direct firing over the tops of the pans, as shown in Fig. 17, the last in this paper.

Objections are raised to this type of plant, on the ground that a certain definite percentage of monohydrated acid is necessarily lost in the current of intensely heated and rapidly moving flame-products and air thrown over the surface. This surely need not occur where, by gas-firing, the products of complete combustion are quietly and evenly held in proximity to the upper surface of the pans until saturation takes place. By this method of firing it will be seen that the outlets for the vapors can be controlled absolutely, up to any definite point of saturation.

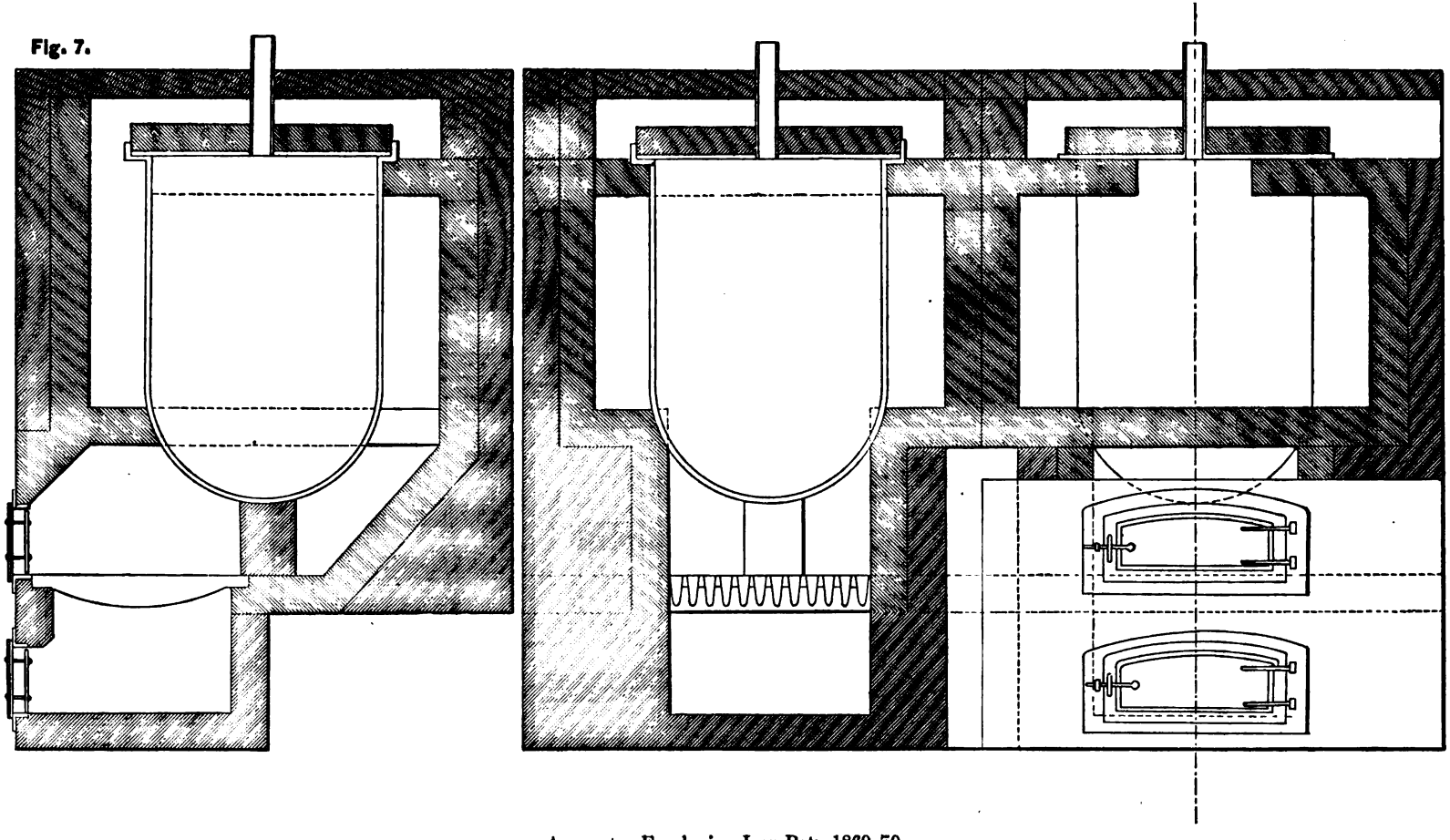
Such regulation is not as difficult as it looks at first glance, and it offers an absolute degree of heat at all points in the line of pan-surfaces, and at all times alike, without excess of air currents. A failure by reason of absence of practical skill in application should not condemn a process of so great promise.

The employment of superheated steam, favorably operated in many of the Continental factories, is practically an unknown process in this country. Under proper conditions, there is much to be said in its favor, and another decade may give us examples worthy of mention.

Following the pans, many forms of apparatus have been constructed to accord with local requirements, and of these this paper will principally treat.

Figs. 1 and 2 show a type of furnace used throughout the world

Fig. 7.



Apparatus Employing Iron Pots, 1869–70.

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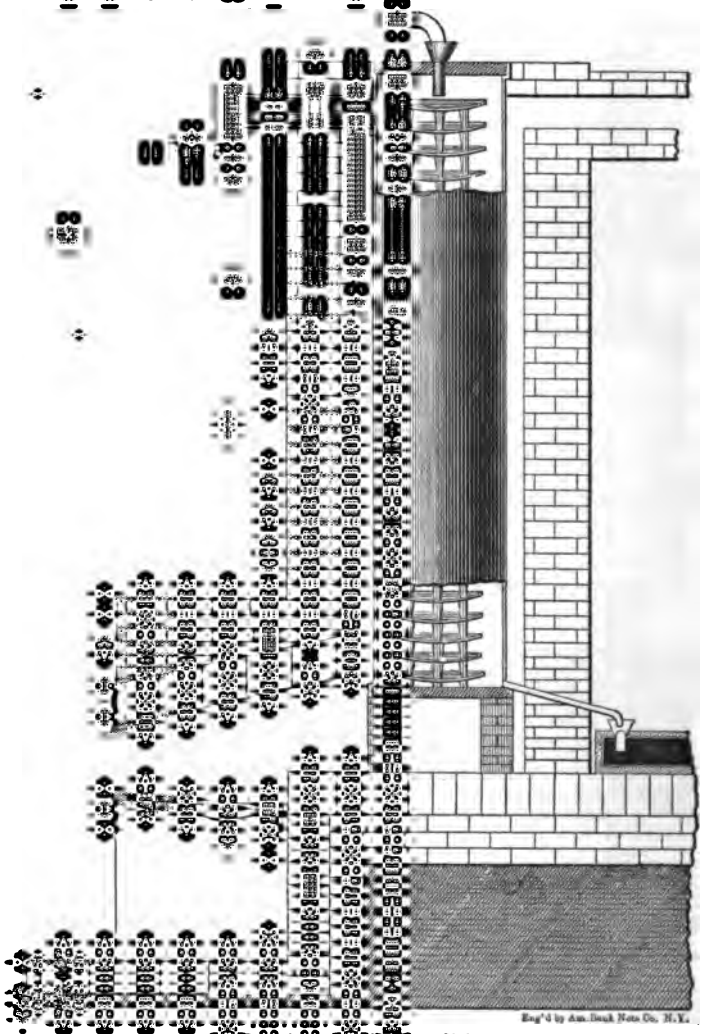

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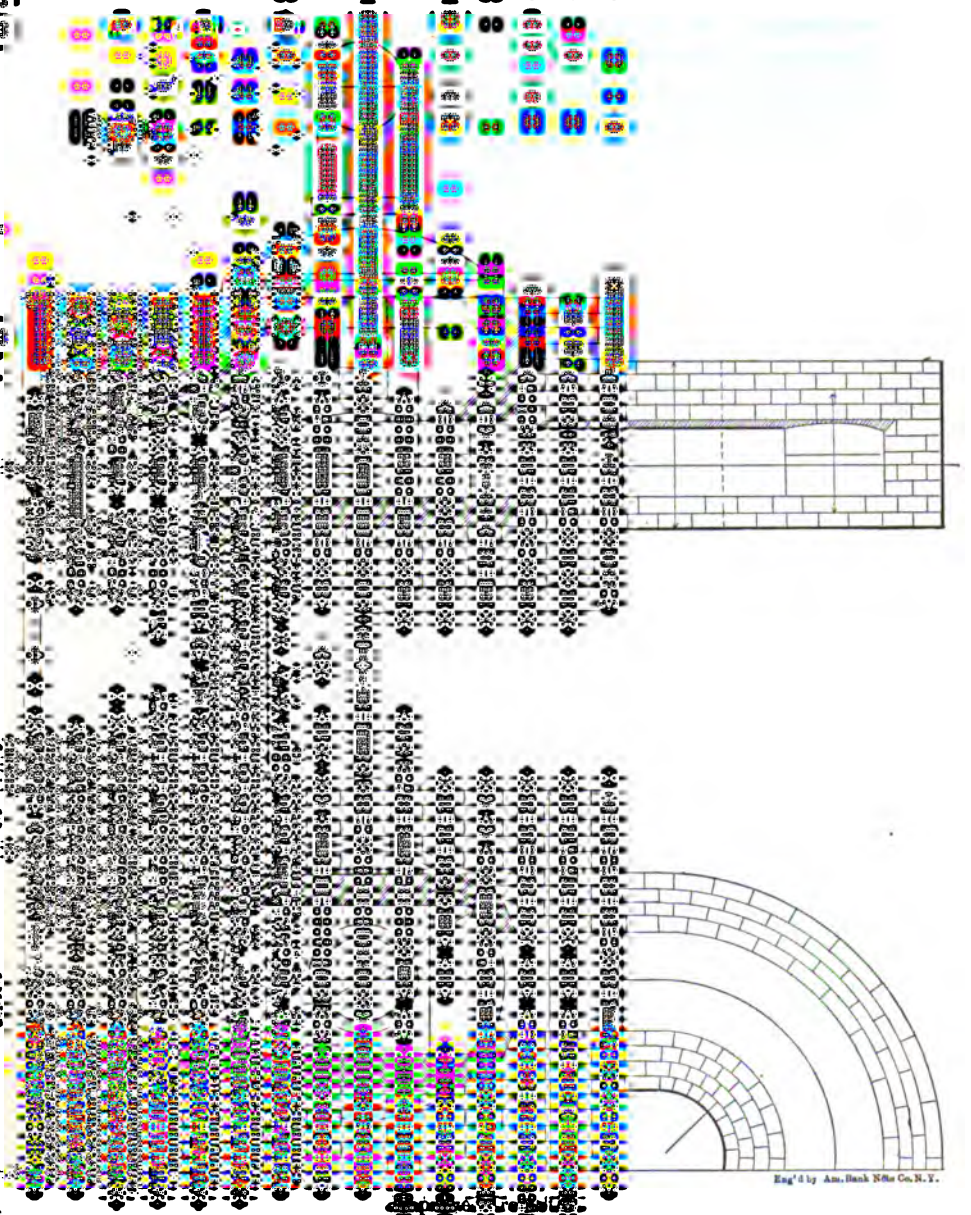

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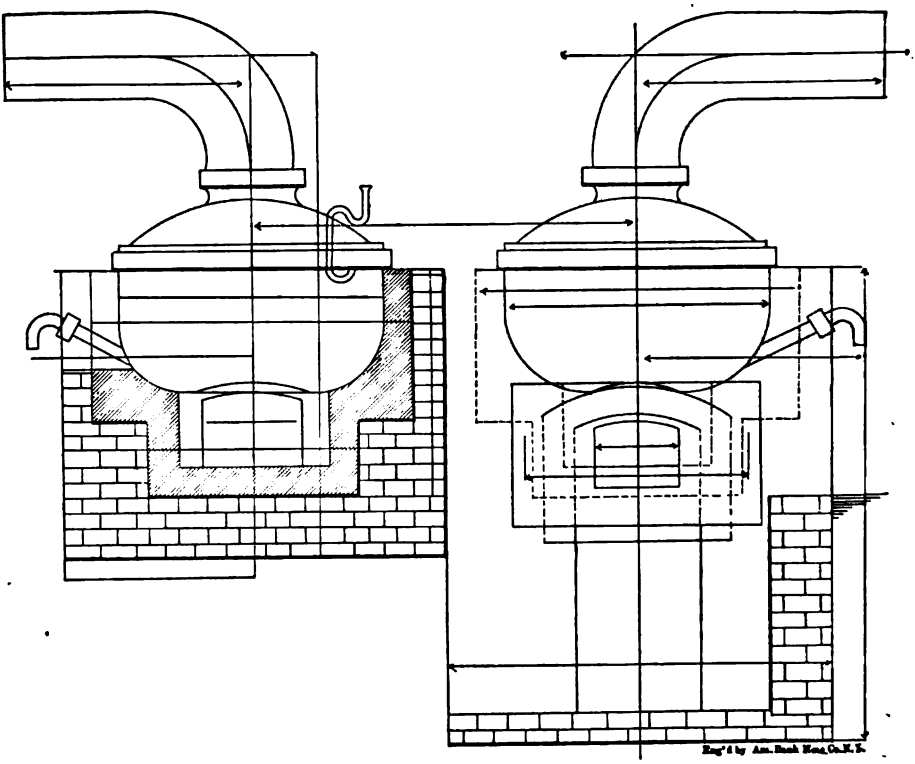
Fig. 3 is a modification of the above, also in use largely at the time mentioned. An iron pot is built into and made a part of the furnace proper, holding the glass retort on a sand-bed intact from fire or draught. At the lowest part of this pot a small cast-iron pipe (not shown) extended through the furnace-walls, and carried off any waste acid from broken glasses into reservoirs of stone or lead which were conveniently placed for that service. The simple construction

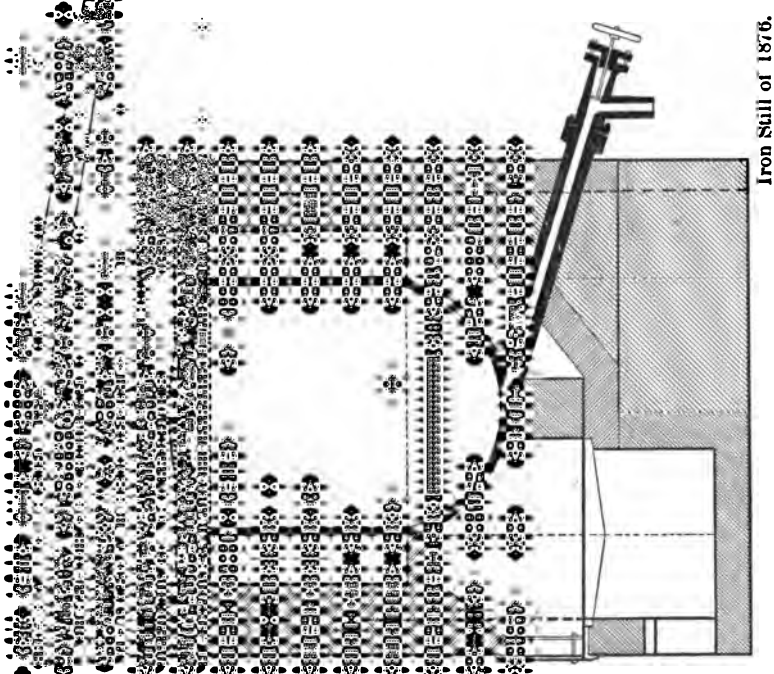
FIG. 9 *a*.



Section of Fig. 9.

of this furnace recommends it; but the protection to the glass which, at first glance, the iron pot seems to give, is not conceded by practical men, as variations of temperature and ordinary causes affecting the one disastrously seem also to affect the other, and the intense heat required to be carried under such pots destroys or deteriorates the iron very quickly, as is well known. My notes, taken over a period of several years, prove that with careful firing

se in pots, and that with either furnace.



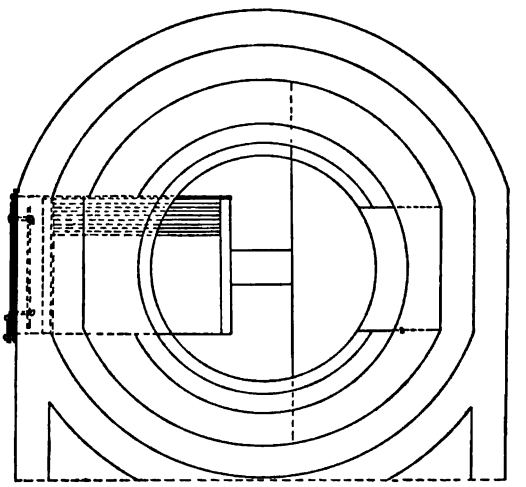
Iron Still of 1876.

e, clean, commercial is cheap as regards small process can be

to-day; but location must be understood to govern costs always. In a favorable location, with cheap fuels, first-rate glass, and average labor, a small factory can hold its own against any newer plants, except the very latest; and we now see works of this class operating satisfactorily in connection with other industries.

Extra demands (war demands, we termed them) for chemicals of all kinds, stimulated the trade to the erection of new works in this country, the majority of trained men for their management coming from Europe. Naturally, the Old-World practice of concentration became firmly rooted here (though materials and apparatus were vastly inferior), and it would be deemed extravagant to now recite the difficulties, losses, and failures our American chemists have met during

FIG. 10 *a*.

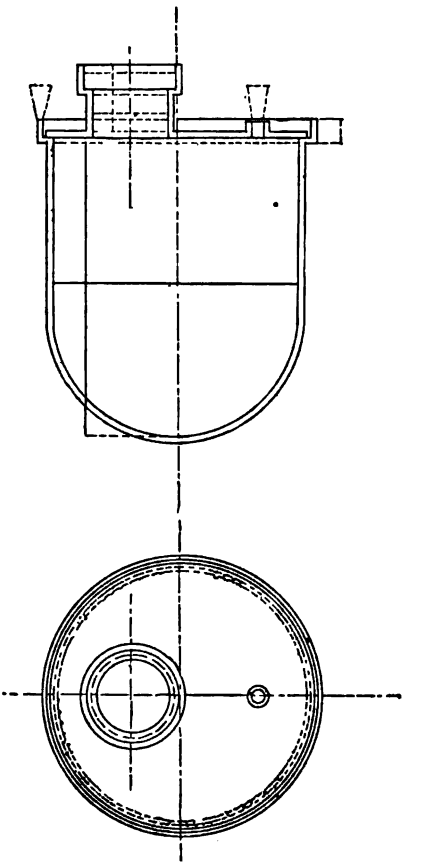


Plan of Fig. 10.

the years that have passed, in arriving at their present position. Notwithstanding the heavy duties, excessive freights, and losses in transit, it was absolutely necessary for many years to import foreign glasses or retorts; and the single item of cost represented by this article was frightfully large. An urgent necessity for lessening this expense led to experiments of all kinds; but those having reference to or approaching the continuous processes, introduced into European factories about that time, received the greatest attention. One of the drawbacks to the style of construction in vogue twenty years ago, was the floor-space required, and objection was met by a new class of apparatus devised by the chemists of Europe, and composed

of sheet-platinum, which so enormously increased the output, reduced the cost of concentration, and simplified the working as to meet with immediate recognition. In the manufacture of anilines, nitro-compounds, etc., several industries dependent upon cheap acids were brought into life.* Such apparatus was unnecessary in

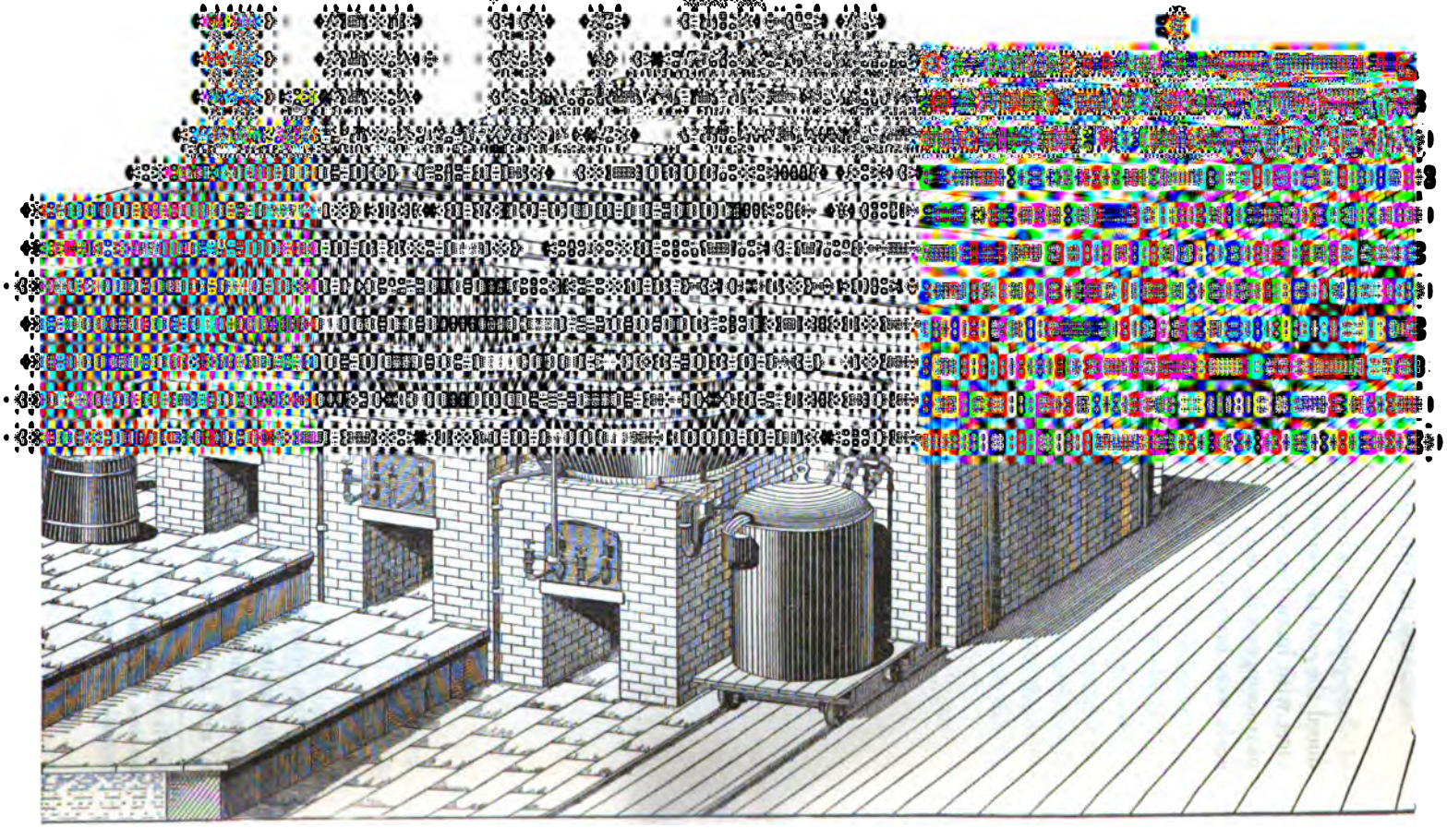
FIG. 10 *b*.



Section and Plan of Pot in Fig. 10.

America prior to the year 1870, the market for commercial acids being limited up to that time, and easily supplied by the types of furnaces shown in Figs. 1, 2 and 3. About that time, however, the oil and fertilizer trades had grown to proportions sufficient to absorb

* It may be said that the cheapening of platinum and its common use date from the time those chemical works made demands for it in quantities sufficient for concentration processes.



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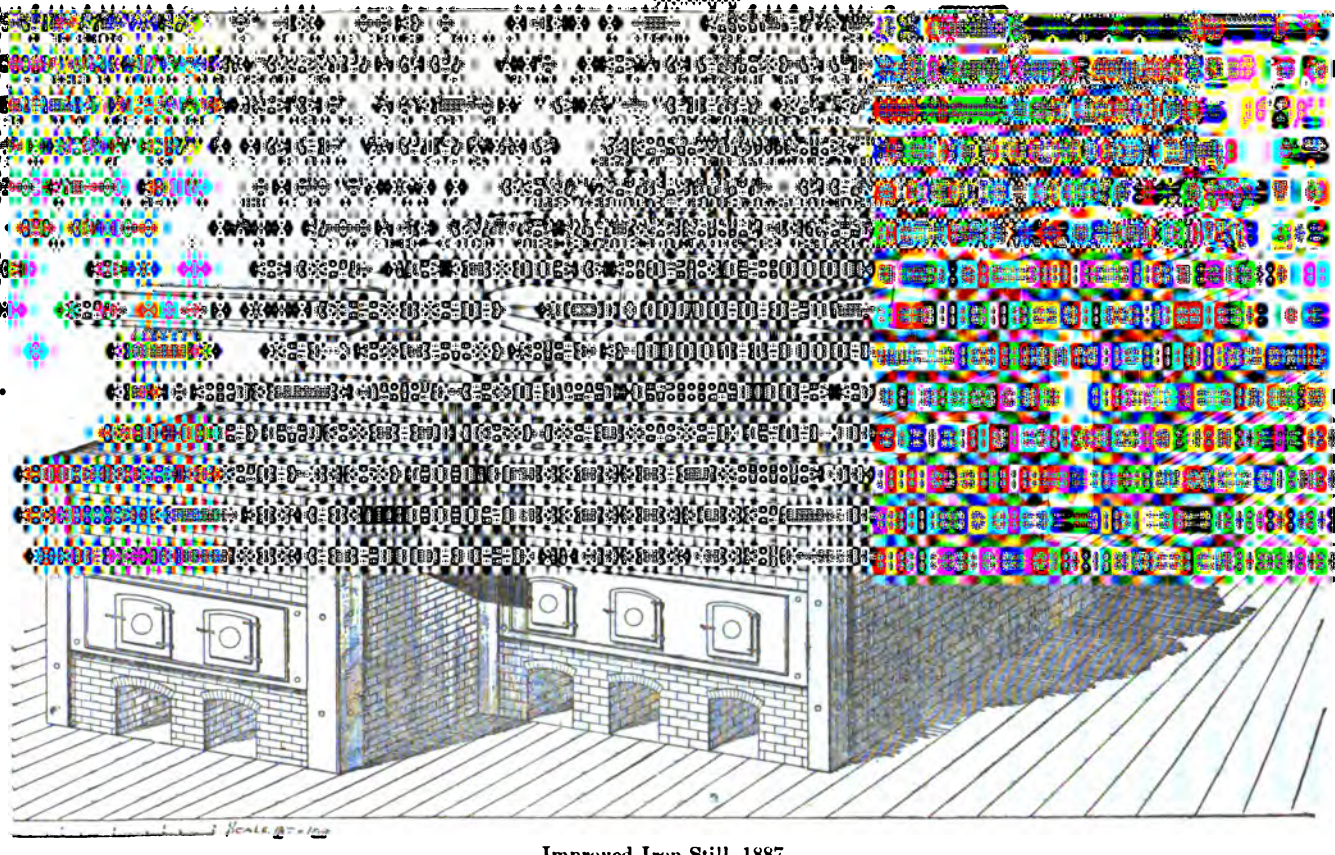
too uncertain in daily use; and this process was only maintained continuously when 5-gallon pear-shaped retorts were employed. The process will be seen to be theoretically correct, inasmuch as the strong acid is drawn from the bottom of the retort as fast as concentrated, and the weaker acids occupy their natural position on the surface. While never applicable for extensive service, yet the process can be made useful for several purposes if fired with gas.

Fig. 5 shows a continuous process, originating in Brooklyn, N. Y., which the writer put into use to replace the one last spoken of. It was in very general use from the year 1869, and can yet be found in several of the smaller works. By this process, the pan-acid (60° B.) runs continuously into the first glass or retort through a funnel which is inserted in an opening in the upper part of the glass. Opposite and below this opening for the weaker acid, a lip or spout is attached to and made, by blow-pipe work, a part of the retort, through which lip the partially concentrated acid runs to retort No. 2, and thence to Nos. 3 and 4 of the series,—a loosely-attached glass tube forming the connection from spout to funnel. The completed product (65° to 65.5° B.) is cooled in leaden pipes or troughs, set in running water, and from such coolers is taken to suitable storage-tanks. The retorts are set in cast-iron saucers on a bed of clean dry sand, free from all organic matters. Other details are deemed unnecessary, as plants of this kind in operation can be studied by those interested.

It will be noted at a glance that this process depends entirely upon surface evaporation, yet withdraws its concentrated product from the surface-line, the stronger acids being held below the discharge-line, and all sediments which are not violently ejected by the boiling must adhere to the sides and bottom of the retorts. As a process for continously producing acid of strength not over 65° B., and where purity is essential, it is far ahead of any known glass-retort plant, but the principle is in every way wrong for the manufacture of acid of greater densities. A gravity of 65° B. on the surface of retort No. 4 means an acid of 66° to 66.3° gravity on the bottom; and at these densities the sediments which are precipitated, and the violence of the ebullition of such liquids, mean sure destruction to fragile glass. Up to 65° B., several metals, such as platinum and gold, would last indefinitely for concentrating service, unless the acids were strongly charged with arsenic, etc.; but beyond 65° B. all the metals generally found in solution (lead, iron, sulphur, etc.), are



Improved Iron Still, 1887.

rapidly precipitated and adhere to the pan or retort-bottoms after the manner of scale on the crown-sheet of a boiler, until such time as something gives way.*

Fig. 6 shows a process aimed to accomplish like results with the previous examples, porcelain dishes and connection-pipes taking the place of glass, and the porcelain, carefully annealed, being depended upon to withstand the variable heats, the shocks of the boiling, and the action of the sediments. Although this and several ingenious devices of more or less merit have been brought to notice, none of them have been commercially successful, or deserve particular notice.

In this connection mention might be made of the several attempts to utilize the heat from the burner gases, but as their sole aim is to act in the manner of the pan-evaporators, *i.e.*, to rid the chamber-acid of a certain percentage of its water, the details of the processes or illustrations of plants are not thought to be within the scope of this article.

The refrigerating process of Prof. A. A. Hayes, of Boston, Mass., never has received the attention due to so eminently satisfactory and simple method of concentration, but we may expect a development of the idea to a practical success, by the newer generation of chemical managers—the easily controlled machinery for artificially producing a zero-temperature being employed.

One of the heaviest items of cost for many years was that of transportation of acids from the existing sea-board works, to meet the demands from new industries springing up in the interior. So long as the major part of their product was consumed near at hand, the glass carboys of the revered age of alchemy, with all their cost, bulkiness, and weakness, were made to do duty for all carrying purposes, and still continue to serve a large share of the trade. But the demand for new forms of explosives (nitro-compounds) developed a trade among new men, remote and often entirely away from all rail- or water-routes—a trade, moreover, too large for handling

* In this place it may not be amiss to mention the losses which take place during the operation of concentration of acids in retorts, from the oxidation of sulphur alone, losses which do not seem to be fully understood but are liable, if not counted, to upset all calculations. The well-known reaction between pure sulphur and sulphuric acid when heated will explain this. Thus: $S + 2H_2SO_4 = 3SO_2 + 2H_2O$. All organic substances exert a more or less destructive oxidizing effect, but this example is sufficient to show a most annoying loss brought about by the common practice of allowing the pans, stills, retorts and connections to accumulate these foreign substances.

Modern Iron Still.

in carboys. Necessity brought about an opportune discovery, viz.: that iron vessels, in the shape of drums, used for the carriage of glycerine from foreign countries, or iron tanks, such as are generally used for the transportation of oil in bulk, were in every way suitable for storing and carrying acid. The advantages accruing to outlying industries by this discovery can only be appreciated by those familiar with the trade for years past. It is matter of history that within the time I mention, prices on acids have been lowered from 50 to 75 per cent., no small share of which is due to radical changes of this character; and although it is not to be supposed that any one cause has cheapened acids to present rates, yet this item of transportation in bulk, in vessels of so common material as iron, has enabled large consumers to so cover the supply points as to virtually free themselves from local dictation. The very low prices which are general throughout the United States are due, in my opinion, to the action of a few strong companies who have solved the problem of the exact cost of production, and have the daring to threaten the main competing points by their ability to command a low transportation rate. That they manufacture so little, comparatively, yet dictate the prices for their enormous consumption of acids, is one of the New World ways of doing things so beneficial to the general public as to create as yet no dissatisfaction at corporate action.*

This employment of iron vessels under such varying conditions, has opened up a new avenue for that metal, viz.: for use in place of glass or platinum in concentrating apparatus. Experiments to perfect the details in this direction have been conducted almost wholly by sea-board works, handicapped by locality and forced to reduce costs in manufacture to meet competition from more modern works erected at points of consumption.

One of the earliest examples of this use of iron, is shown in Fig. 7; a type of pot in general use for the manufacture of nitric acid during the last 50 years. It was about 1869–70 that experiments were made with this pot, in the manner of the glass-retort procedure, viz.: filling the pot full of pan-acid, and boiling until sufficiently con-

* In assisting us to understand why a comparatively new country, consuming so little of its own mine products, can thus hold the price of a leading commercial article at so low figures, it may be well to remember that the largest consumers of acid are the oil industry and the alkali trade, each in its own sphere financially able to make prices, and each, in a widely different pathway, subject to laws which have brought those prices to almost identical figures.

Improved Still for Strong and Pure Acid.

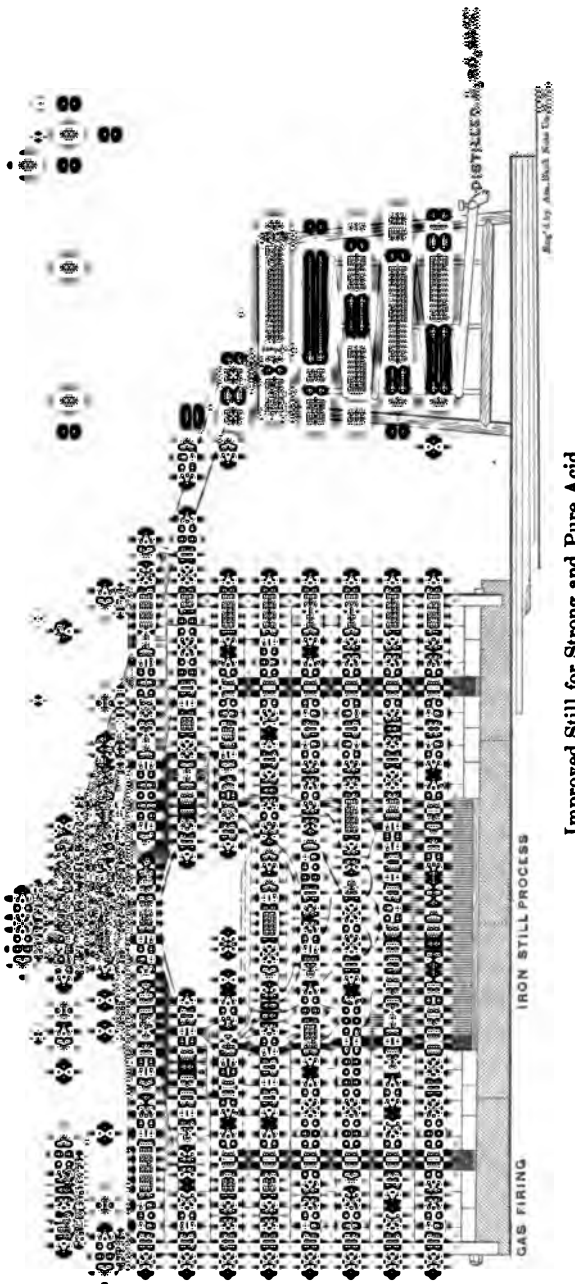
centrated. Those who have handled the stronger nitrous acids will appreciate the difficulties, and will not wonder at the slow progress made in overcoming the mechanical drawbacks, nor that discouragements in properly protecting the surfaces of iron next the outlet, and the connecting pipes which carried off the vapors, finally led to cessation of the trials. In my drawing of Fig. 7, the pot is represented as built entirely within the walls of the fire-place, but the earlier experiments were with naked tops, therefore subject to greater corrosive action than the sketch would indicate.

Another experiment attempted about this time, as an outcome of the former test, is illustrated in Fig. 8. A cast-iron cylinder was built into a chimney-like furnace-space, so as to confine the heat closely to the metal from bottom to top. The cylinder was closed with slabs of non-corrosive stone (such as are employed for the fronts of nitric stills), and the space inside was nearly filled with saucers of porcelain, set one upon the other from the bottom. Pan-acid ran into the top saucer and spread itself thinly over the sloping bottoms of the saucers until it was taken from the exit-pipe in a sufficiently concentrated condition (66° B.). The outlet for chimney-gases, the care of the concentrated vitriol, and other details, will be understood.

It is plain that a cylinder evenly and strongly heated at all times means an inner radiation which will very rapidly drive off all the water from the thinly flowing liquid. So great an evaporative surface with so small a construction-cost, promised a most successful result; but difficulties not connected with the work itself, though involving a change of base for the experimenter, brought the experiment to a close. It is here noted to show how closely success and failure are allied. In the hand of a practical man, allowed the privilege of experimenting, this apparatus should have been made an unquestionable success, the germ of what we now know to be successful working being certainly shown in this construction.

It is unnecessary to detail many of the changes which followed the above plant, the next decided step being shown in Figs. 9 and 9*a*. This design also closely follows the successful platinum apparatus of that time, as this reproduction of the original drawing will prove. The action of this form of still in service will readily be understood, the construction being modified from time to time, so that the still was finally enclosed in the furnace, the fire surrounding it completely.

Several of the most important improvements credited to the last



Improved Still for Strong and Pure Acid.

decade have been made by the engineers who devised this last-mentioned still and the one which immediately follows. Fortunately for the literature of the subject, they are as willing to impart their knowledge—giving to the student the benefit of such details as are thought interesting—as assuredly successful originators always are, a commentary on the world of copyists, who manage to secure ideas only when they are discarded by original thinkers, and then attempt to place them under lock and key as original matter.

Figs. 10, 10*a* and 10*b* illustrate the outcome of much experimenting from 1870 to 1880. Made from the original without change, they will clearly prove, to those but now commencing the use of iron, the perfect knowledge of the subject which the designer possessed. Except constructively, there is little room for improvement in this apparatus, and the successfully working plants of to-day will be found to closely follow in principle, if not in form, this design of a still worked in 1876. In the light of well-known facts, it is difficult to understand why this virtually perfected principle was not adopted by the trade generally, or at least added to the glass and platinum apparatus in all large works, especially where pyrites was used.

It is not uncommon to find an investment of $20,000 to $30,000 in platinum stills, pans, etc., in a plant for an output of thirty to forty tons commercial vitriol daily—several of our American works being equal to this demand. Such an outlay, with attendant wear and tear, interest, and chances of destruction from various causes, can of course be borne by but few companies; and the actual cost of running such a plant, though conceded to be very great (some authorities putting it at not less than twenty per cent. per year on the investment), has never been published by those alone competent to give figures extending over a term of years.

Several causes, unnecessary to discuss, have combined to bring these foreign-made stills into use in this country, yet in Europe it is an open question, considering all things, whether the cost of concentration is much less in them than in glasses. Investigations into minor details of costs, losses, etc., are more systematically pursued by the chemists of the Old World, and a primary factor, acknowledged by them in this connection, has been, as yet, very little understood by owners of works in America. I refer to the very rapid destruction of platinum by nitrogen compounds—a common trouble, too often credited to base elements in low-grade brimstone or pyrites ores. Chemists are fully aware how difficult it is to rid sulphuric

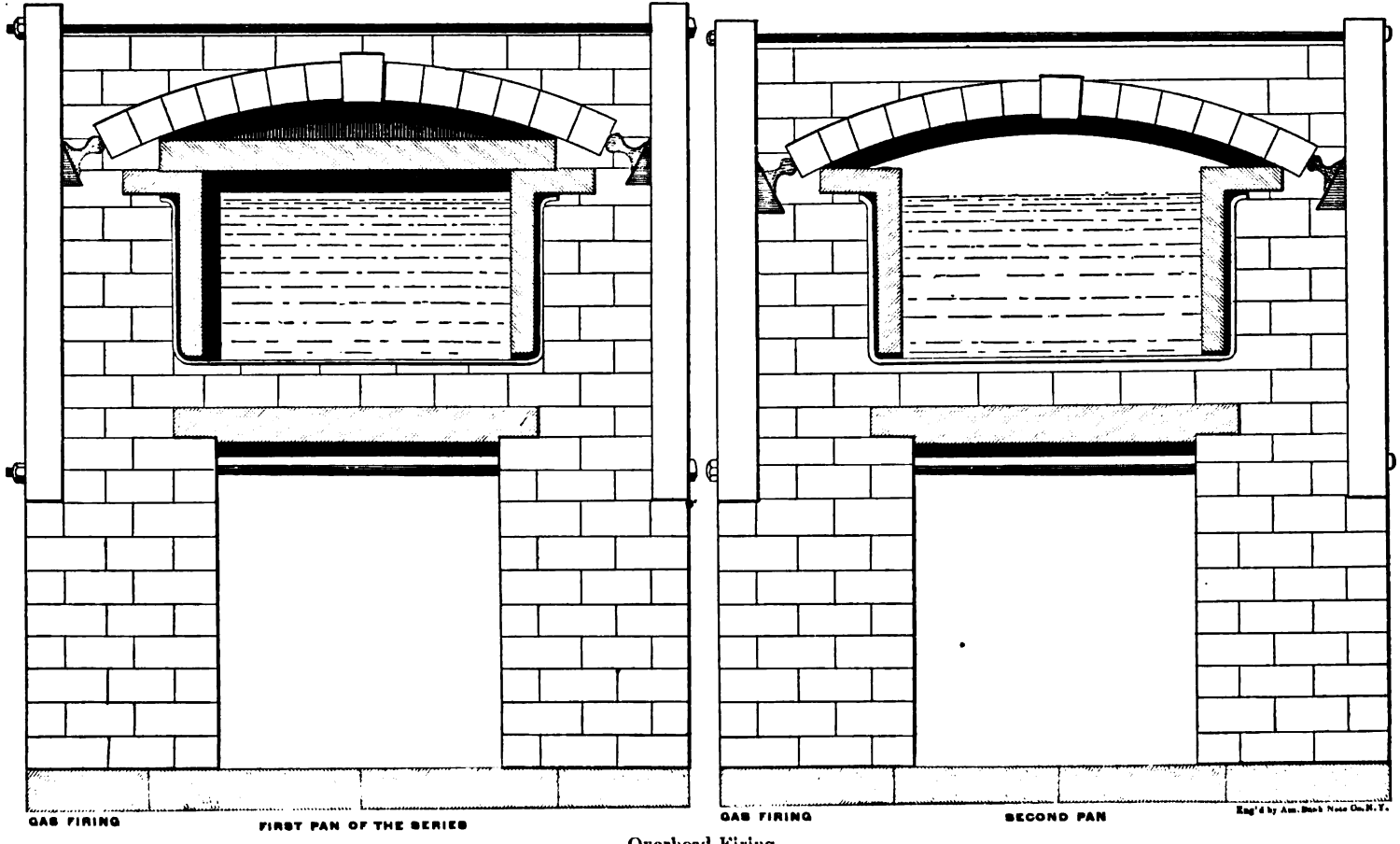
acid from any small percentage of nitrous acid, and that after many costly experiments the cautious use of ammonium sulphate is now wholly relied upon for this purpose. Considering the few works making use of any process for neutralizing nitrogen compounds, there can be no question but that glass retorts are preferable to platinum alone for general service—certainly so where pyrites containing arsenic is used, and where the demand is for 66° B. acid alone.

These foreign-made stills and pans of varying dimensions, are too well known and have been so fully advertised by the patentees as to need no mention in this paper; but, as illustrating the most advanced type of shallow-basin platinum stills and their accessories, I am enabled to present, in Fig. 11, the largest and most successful producing plant in the United States.

For obvious reasons, the details cannot be given; but the plant, as shown, has a capacity of 500 carboys—say, 80,000 pounds—daily, and the fuel and labor cost are the lowest yet reported. Outside of the deterioration of plant, or waste of platinum, this construction, with gas-firing, guarantees the best results yet known for commercial acids of 66° Baumé. As suggested in my remarks on the glass retorts of Fig. 5, the large cost of concentration, whether in fuel, labor or deterioration of metal, is expended while increasing the specific gravity of the acid from 65° B. to 66.4° B.; but if the platinum is not called upon to concentrate the acid above 65° B., the losses on any part, or upon the entire plant, become a matter of easy computation.

This brings me to consider the advisability of combining the work of the two metals; namely, the addition of one or two iron stills, of late type, to the existing platinum plants. In this connection I am aware that no successes, on a large and practical scale, are as yet publicly credited to the iron-still plants. Yet it is nevertheless true that iron stills, which cost less than $100 each, are placed alongside platinum-basin pans; are doing the final concentration of acids of greater density than 65° B., without appreciable attack from arsenates, nitrates, etc.; are being driven at any required pace, without material deterioration; are handled as easily as platinum in every way; are working at less cost, in proportion, and have been thus working for several years. I am aware that the greatest objection so far brought against the use of iron for this purpose, has been the exceedingly short life of the metal in service, and that much, if not all of the wear comes upon the easily corroded exposed connection-joints. These minor difficulties, rather than condemning the process, should

FIG. 17.



Overhead Firing.

lead to increased care and attention to details of every-day working, without which even the glass-retort plants would be failures. These problems have all been thoroughly and practically solved by a few of the progressive engineers of this country, and it is now a fact that stills which cost $25 can be run on an average for four months, while, in the earlier stages of the experiments, we were satisfied with a cost of $100 and a life of two weeks.

New as the ideas undoubtedly are, the drawings herewith will have due weight among practical men, as every detail of the setting, removal in case of breakage, etc., will be understood at a glance; the elaboration of the details being more for the study of those who have had an experience with glass alone. Fig. 12 shows a modified and perfected form of iron still, two of which type are connected direct to pans; and taking such pan acid at 63° B., are turning out 12,000 pounds of 98 per cent. oil of vitriol each day. Fig. 13 shows an entirely different construction, a modification of which, and smaller, is running near New York and connected to platinum, is turning out nearly 30,000 pounds of 98 per cent. oil of vitriol daily. Fig. 14 shows still another modification, one of the latest and most successful types now at work. The simplicity and inexpensive construction of this apparatus commend it on sight; and, with gas-firing, the plant is a long way in advance of any made known as yet. By this entirely new method of working, the flow of acid from pans to stills is continuous; the evaporation from the surface of the still proceeds rapidly and quietly; the sediments, from whatever source, are swept along and out of the still in the constant flow of a small current of strong acid; and, taking temperature and density into account, are precipitated in regular order in the larger space of the outside boot, where they remain until removed by the basin-shaped ladle.

These two stills, gas-fired, will produce about 40,000 pounds of 98 per cent. oil of vitriol daily, when fed with hot acid of 92.5 per cent. strength.

Without going too deeply into facts which properly belong to factory owners, it is sufficient to say generally, that cast-iron has come to stay, with a value for chemical purposes as yet little understood, but which the world must naturally profit by; a value due almost wholly to investigations, experiments and successes of Americans; a value sure to be hereafter acknowledged in the making-up of cost-sheets on sulphuric acid. This practical use of the metal marks a change as worthy of recognition as the Gay-Lussac column,

or the Glover tower, in their respective spheres—each of them most radical changes in the manufacture of this essential product.

By this use of iron, and this method of handling the contaminated acids produced from ores so abundant in our country, all the difficulties heretofore met in the final preparation of strong sulphuric acids are overcome at a bound. Local circumstances will naturally call for modifications in the use of cast-iron stills; but the main difficulties are already overcome, and it is not unlikely that the prejudices against the working of pyritic ores may, by this new use of iron, be dissipated, as has been the case with prejudice against the towers.

In conclusion, it may be noted that within a short time, several of the European factories, consumers of pyrites containing appreciable quantities of arsenic, have been called upon to manufacture an acid of guaranteed purity and strength, absolutely free from lead, selenium, arsenic, etc., which are invariably found in commercial acids. They are now filling this demand by distilling the ordinary run of 66° B. acids; several continuous-pan processes having been devised for that purpose. The distillate is condensed in glass, platinum, pottery coils, worms or pipes—no specially new features being noted which call for mention. In this process, the resulting stronger acids are removed at intervals, by stoppage of the plant.

This manufacture of virtually a new article in the trade (an acid of regular strength, and pure) will become general throughout the world in course of a few years; and we may expect to have presented many devices, of a more or less complicated nature, to accomplish what is, after all, only a simple problem of distillation, which has heretofore been attended with difficulties solely on account of the tendency of the metals or minerals held in solution, to cake or cement on the bottom of the stills, as explained.

It will be seen that nothing can be simpler or more effective for this service than the pot shown in Fig. 10, modified to the present perfected stills as shown in Figs. 15 and 16.

Here the absolute control of the flow of acids enables the operator to accomplish both results, namely, the manufacture of 93 per cent. and of 98 per cent. oil of vitriol simultaneously, or either of them at will.

Fig. 17, which has been placed last to separate it from the series represented by all the rest, is sufficiently explained on page 7.

MODE OF DEPOSITION OF THE IRON-ORES OF THE MENOMINEE RANGE, MICHIGAN.

BY JOHN FULTON, JOHNSTOWN, PA.

(Duluth Meeting, July, 1887.)

THE Menominee range is one of the four great mining belts that flank the western prongs of Lake Superior. It has received its name from its location along the north side of the Menominee river, which at this place flows between the States of Michigan and Wisconsin. The term "range" is derived from the elevated ridge, extending in a westward direction, and along the southern flank of which the iron-ore mines are located.

This range is some 200 to 300 feet high above the general level of the adjacent country, and is made prominent by its surroundings of low, swampy lands. Its whole length is about 27 miles, and it is intersected, at intervals along its course, by gaps which are threaded by streams.

The geology of the Menominee range is peculiarly interesting. It is composed of three distinct groups of Huronian rocks. The supporting group consists of light-colored limestone beds of a crystalline, siliceous character, called the "Norway limestone belt."

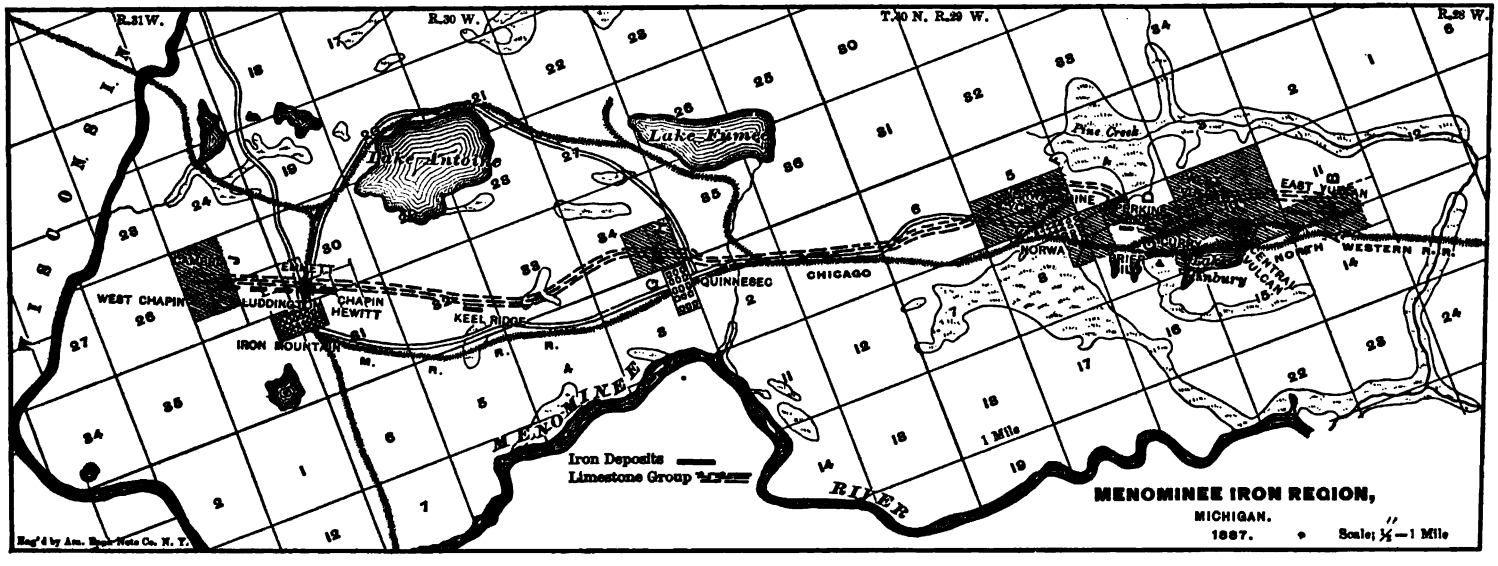
The map, Fig. 1, and the section, Fig. 2, will give a general notion of the position and structure of the range. The properties shaded in the map are those of the Penn Iron Mining Company, which is the Michigan representative of the Cambria Iron Company of Johnstown, Pennsylvania.

This belt is at least 1200 feet thick. It outcrops boldly at many localities along the range, especially north of the Norway, Quinnesec and Chapin mines. At Quinnesec it can be studied readily.

The next, or flanking group, which is estimated at a thickness of 1000 feet, is called the Quinnesec ore-formation. It consists, in the portion next to the limestone, of siliceous or jasper slates, largely impregnated with iron oxide. These are succeeded by argillaceous hydro-mica black and flesh-colored slates. This formation embraces the deposits of the iron-ores.

The third group consists of a series of dark gray, slaty or schistose beds, with occasional quartzose bands, having a thickness of 2000 feet. It is called the Lake Hanbury slate group.

FIG. 1.



Sketch-Map showing the course of the Menominee Range.

These three distinct groups of rocks, forming the range, succeed each other conformably, dipping at high angles to the south in the portion of the range east of Quinnesec; west of this they dip northward.

The order of superposition of these three groups of rocks affords no evidence of their relative ages; for the limestone is the floor of the series east of Quinnesec, while west of this it is the uppermost. As no fossil remains have yet been found to determine the relative ages of these formations, their order in time will require to be determined by the geologist, aided by the microscopist.

The crest of the ridge is capped at frequent localities with rather massive, horizontal layers of drab and buff Potsdam sandstone, from 10 to 300 feet thick.

The geology, although not complex, yet, by reason of the frequent and violent flexures and strangely postured dips, becomes very perplexing, even under long study. This is intensified by the absence of the fossil remains that contribute so much to the ready identification of the younger families of rocks. Very much of the country is hidden by gravel and sand-beds, by the drift and wash generally, and by the cappings of the Potsdam sandstone.

The observer can hardly fail to be immediately and deeply impressed with the sedimentary character of all these rocks. At the Quinnesec mine the iron-ore is finely stratified in leaves almost as thin as paper. The foot-wall jasper slates show an equally attenuated bedding. The limestone and the sandstone alone show thickened bedding. The interval of time between the formations of the Huronian rocks, pitched, flexed and denuded as they now are found, and the quiet, soft, horizontal beds of the Potsdam sandstone—the old sea-bed leveling up the inequalities of the ancient denuded rocks —must have been very great.

It is remarkable that the lower beds of the old-time sea-sand are at many localities brecciated with dark-gray and red jasper slates, showing the swashing and wreck of the older rocks, accompanying its advent. Some of the old-time sea-sand has found a way through cleavage-cracks into the iron-ore, 200 to 300 feet beneath the surface.

Immediately west of the Quinnesec mines a large area of brecciated surface is exposed, which has been polished and grooved by glacial agencies. From the different colors of the pieces of rocks composing this cemented and polished mass, it presents the appearance of mosaic work.

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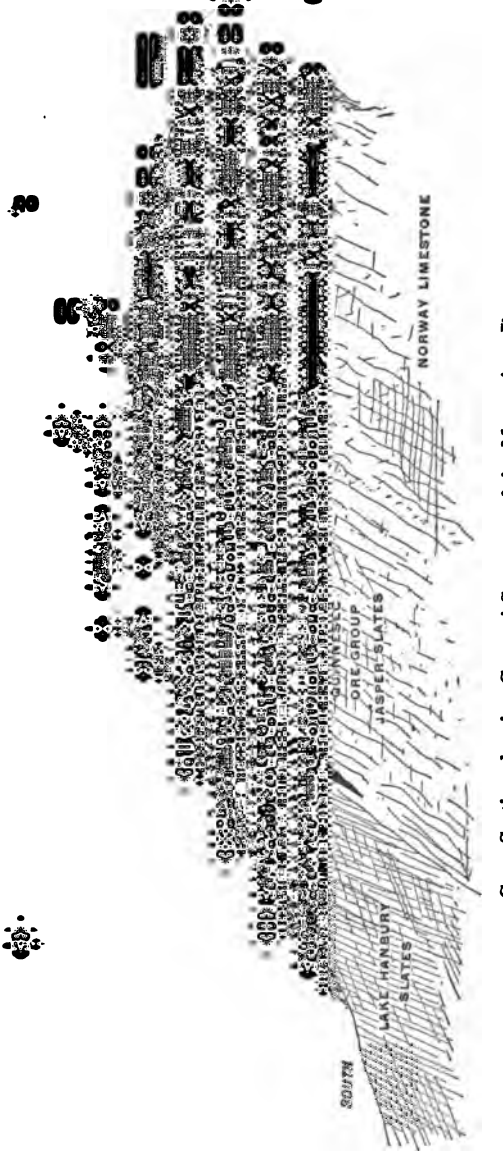

Cross-Section showing General Structure of the Menominee Range.

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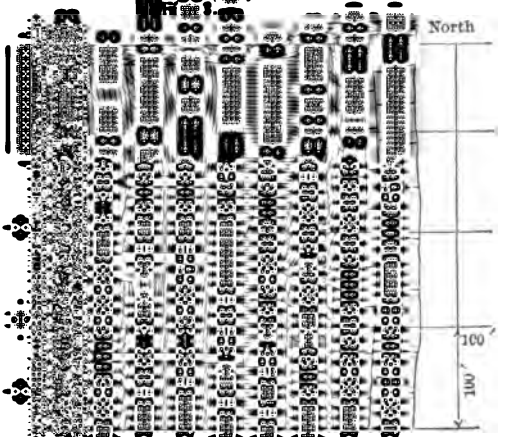

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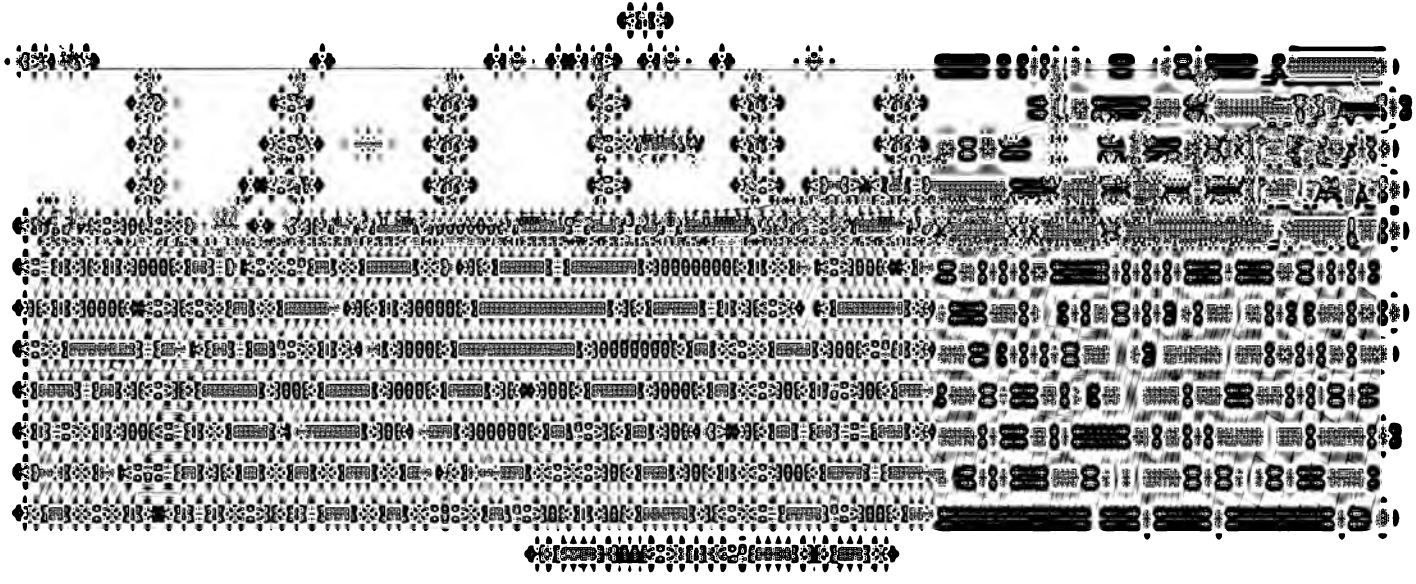

Cyclops and Norwny Mines. Section E F (Fig. 1.)

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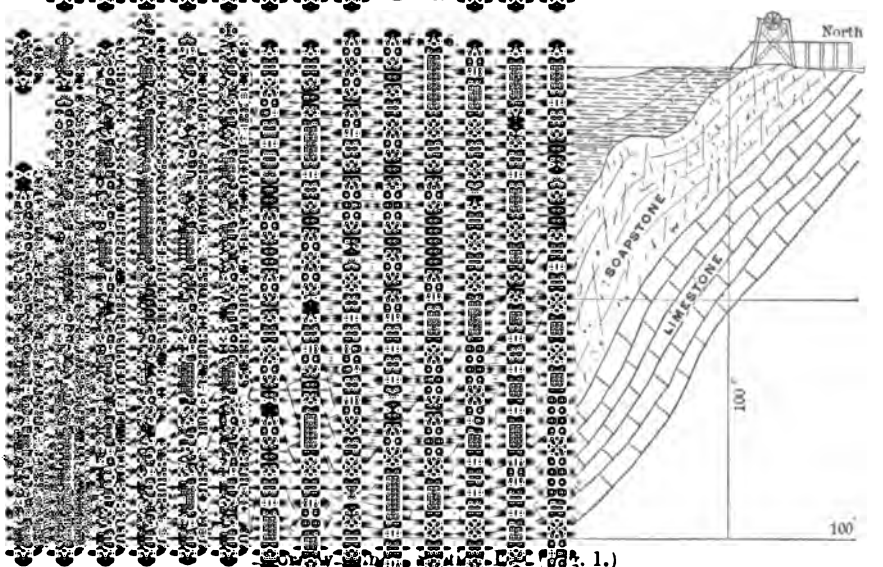

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These pits are 400 feet long and 50 to 70 feet deep, affording over 100,000 tons of excellent soft blue-ore. Fig. 6 shows their general conformation. The ore was evidently carried into them by some eroding current, from the wreck of an original deposit. Just where it came from, or under what conditions it was swashed into these large pits, cannot be determined. It has been considered that the ore in these pits was the enlarged outcrop of deposits that might be traced underneath, but no such trace has yet been found. The example of a pool of ore at East Vulcan illustrates the supposed condition of these Cyclops deposits. But no trace of their relationship to any ore-deposit now existing has been found.

These pits were partially covered by the soft buff-colored Potsdam sandstone in great masses, with quiet and nearly horizontal bedding, resting on the steeply-dipping jasper slates.

The Quinnesec deposit of blue-ore is peculiarly typical of the mode of the iron-ore deposits of this range. It dips 31 degrees to the north and has the limestone for its hanging-wall 150 feet distant. The deposit of ore is 5 to 30 feet thick, and is mainly in the jasper slates.

A remarkable nest of blue-ore which has been found enfolded in the soft Potsdam sandstone is shown, Fig. 7, which also illustrates the mode of transfer of the ore into the Cyclops open pits. This Quinnesec sandstone-enveloped deposit is 30 feet long and 40 feet broad, and has afforded a considerable quantity of soft blue-ore.

A short distance east of the mine a sharp flexure occurs, similar to the Norway, but of much less magnitude. The dip of the range from this westward is generally northward. The Chapin and Ludington mines show this very clearly. The ore-body in the former is 60 to 100 feet thick; in the latter, about 60 feet thick.

Explorations made west of these mines by the Penn Iron Mining Company have developed the relations of four series of rocks composing the range, and also the relative distance of the ore-deposits from the siliceous limestone. A shaft 230 feet deep cuts through the surface-wash (mainly of sand and gravel), thence through the

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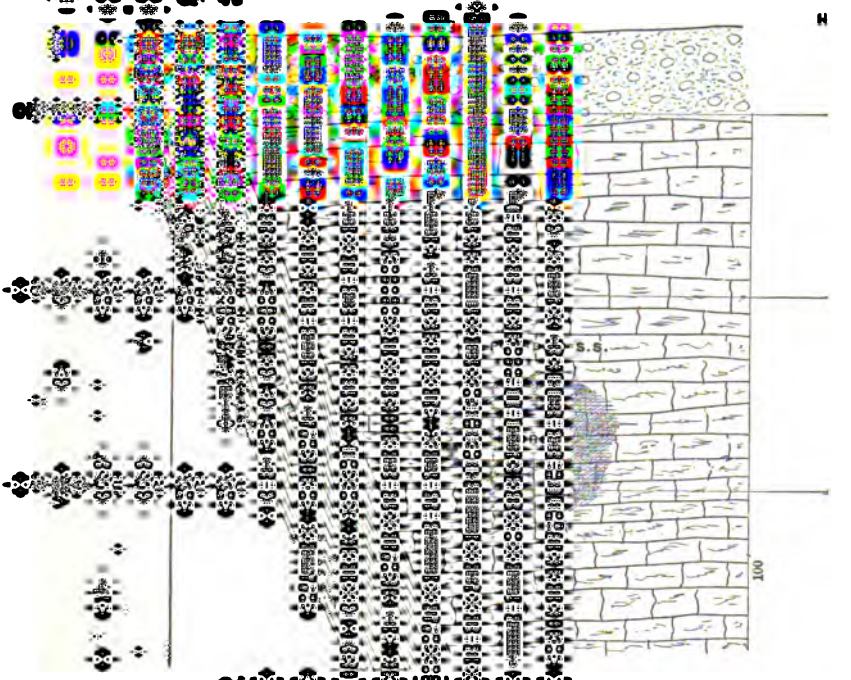

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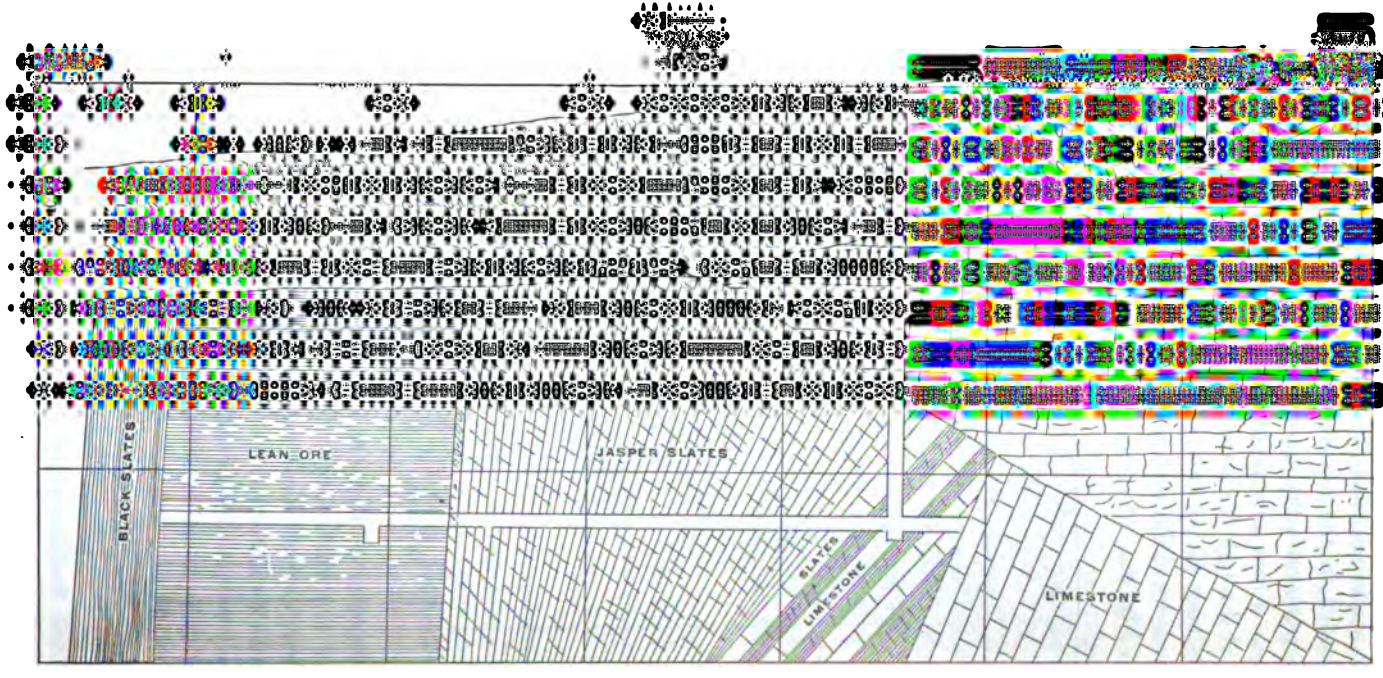

Western Explorations of Penn Iron Co. Section I J (Fig. 1.)

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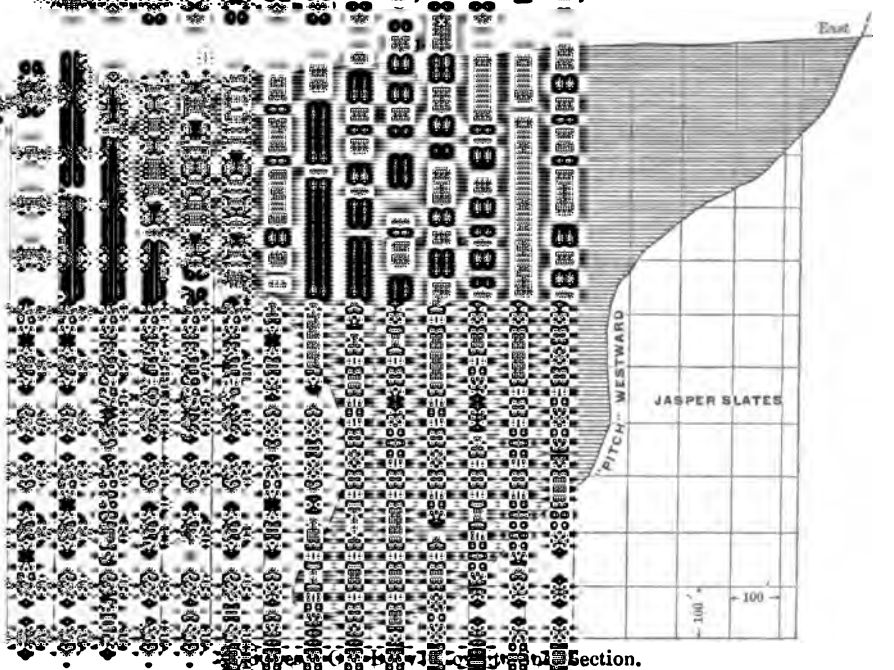

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The following table exhibits the composition of these ores:

Mines.	Met. Iron.	Insoluble Matter.	Phos.
Vulcans	58.75	9.00	.045
Cyclops	61.37	7.33	.015
Norway	53.27	12.02	.036
Quinnesec	64.56	7.69	.037

The shipments from the Menominee range, from its opening in 1877 to the close of the shipping season of 1887, 10 years, amounted to 7,611,998 tons.

This region invites extended and earnest exploration for the further disclosure of the structure of its ore-bearing ranges. It is difficult to accept a continuous monoclinal structure over wide ranges of country. The more evident structure should consist of a series of crust-flexures, repeating the iron-ore measures at intervals, as they rise on the crests or flanks of anticlinal waves.

It is also probable that by further exploration on this range new localities can be discovered inclosing additional lenses of ore.

There are many enigmas in this peculiar and interesting region to be solved. But the laws governing the location, posture and general characteristics of these lenticular deposits of iron-ore are being determined link by link; and, although many difficulties stand in the way—swamps, drift, sandstone,—yet there are places where the groups of ore-bearing measures can be readily explored. The energy and push characteristic of the western miner will develop these and sustain the general productiveness of the range.

NOTE ON A NEW DEVICE FOR OPERATING BLAST-FURNACE CHARGING-BELLS.

BY FRANK CALVIN ROBERTS, C.E., PHILADELPHIA, PA.

(Boston Meeting, February, 1888.)

AN examination of the various devices employed for operating blast-furnace charging-bells has led me to design the arrangement shown in the accompanying figure. In order to contribute to the even distribution of stock, the movement of the bell should be in a vertical line, since any tendency to side-thrust which may be in-

duced operates to the detriment of the even distribution of material. It is also essential that a tight joint should be secured between the hopper-ring and the lip of the bell. This is possible when the bell, in returning after each discharge, seats itself instantly and firmly around its entire base. These requirements render it highly desirable that any tendency to the vibration of the bell should be obviated, and that, to secure proper action, the center of suspension should move in a vertical line. Again, the high temperature to which the bell-rod is, at times, exposed renders it necessary that some simple means should be provided for the adjustment of the bell to its seat.

These requirements are fulfilled in the design illustrated, in which D and E represent the bell-beams pivoted to the eye-beams H, and joined together at their extremities by the links C. From the centers of these links is suspended the bell A, by means of two rods, B. The steam-cylinder G is connected to the bell-beam B, and the weight of the bell is counterbalanced by the weight F, attached to the beam E. The operation of the device is evident. The dropping of the bell causes the centers of the link-pins to travel in arcs of opposite directions—the point of suspension between them maintaining a neutral or vertical line.

The adjustment of the bell to its seat is effected by varying the length of the piston-rod connection to the bell-beam D, by means of a clevis, with adjusting-nuts on the piston-rod above and below the seat of the clevis. This adjustment may also be secured by the insertion of a right-and-left nut (turn-buckle) between the clevis and the piston-rod.

The bell may be easily removed by a slight modification of a familiar device. The spacing of the holes K, K_1, K_2, etc., shown in the bell-rod B, is equal to the stroke of the bell-cylinders. Straps, J, are riveted to each of the supporting beams H, and are of such length that, when the bell is seated, a pin may be inserted into the hole K, with the extremities resting on the straps. The weight of the bell is in this manner supported independently of the beams D and E. The link C may be disconnected from the bell-rod and lowered to a point where its center is opposite the hole K_2; connection being made between the link and the rod, steam is admitted to the top of the cylinder and the bell raised until the hole K_1 is opposite the lower extremity of J, when the insertion of a pin, as before, relieves the beams D and E of the weight of the bell. The operation, as described, is repeated until the bell is lifted clear of the hopper.

Device for Operating a Blast-Furnace Charging-Bell.

This design for operating blast-furnace bells, with the addition of an air-cushion cylinder, connected to the beam E, has been applied to the following furnaces:

Three 18′ x 75′ furnaces for the Alabama and Tennessee Coal and Iron Co., Sheffield, Alabama.

Two 17′ x 75′ furnaces for the Sloss Steel and Iron Co., Birmingham, Alabama.

One 15′ x 65′ furnace for the New York and Perry Coal and Iron Co., Shawnee, Ohio.

Two 12′ x 60′ furnaces for the Nashville Steel, Iron and Charcoal Co., Nashville, Tennessee.

One 12′ x 60′ furnace for the Decatur Land Improvement and Furnace Co., Decatur, Alabama.

MODES OF OCCURRENCE OF PYRITE IN BITUMINOUS COAL.

BY AMOS P. BROWN, E. M., ASSISTANT, GEOLOGICAL SURVEY OF PENNSYLVANIA, PITTSBURGH, PA.

(Boston Meeting, February, 1888.)

PYRITE, the bisulphide of iron, FeS_2, is found more or less in all coal-beds, but, as a rule, in certain definite forms. More than any other impurity, it detracts from the commercial value of a coal-deposit, and yet its presence is so universal that it might almost be said that there is no coal entirely free from pyrite. My observation has been confined to the Pennsylvania bituminous and semi-bituminous coals, and to them, therefore, my remarks will be confined. Certain generalizations which I may make, however, will be found of universal application.

The lowest members of the coal-formation, No. XIII. of the Pennsylvania bituminous region, are nearly always pyritiferous, and in many cases highly so. In Cambria county, for instance, coal-bed A, the Brookville, is locally known as the "Sulphur seam." The Lower Kittanning, in parts of Clearfield county, is so sulphury as to be almost unmarketable, though otherwise it is a strong locomotive coal. In Mercer county the Pardoe bed, so extensively mined there, carries great quantities of sulphur in the form of pyrite, though usually this can be separated in such a way as to detract little from the market-value of the coal. The other coal-beds of the Lower

Productive series all carry more or less pyrite; but it occurs so as to make the coal unmarketable in certain localities only.

The Pittsburgh bed, the only member of the Upper Productive coal-measures which is mined to considerable extent, does not carry much pyrite as a rule; but, on the other hand, what it does contain is almost impossible to separate without washing, a remedy only applicable to coking coals.

Pyrite is found in coal in the following well-marked forms:

1. In nodules or lenticular masses.
2. In defined and persistent stratified bands.
3. In thin flakes and "knife-edges," parallel to the bedding of the coal.
4. In coatings or crusts, lining joints or cavities in the coal.
5. In fine, rounded particles, scattered through the substance of the coal.

As to its constitution, pyrite is either compact (the "hard sulphur" of the miners), or friable (the "soft" or "black sulphur" of the miners). The former occurrence is more common to the lower coal-beds, the latter is the usual form in the upper coal-beds.

1. Nodular or lenticular masses of pyrite are found especially in the lower coals,—the Mercer beds, and the Brookville and Clarion beds. In Mercer county, where these coals are extensively mined, the lenticular masses often attain a width of 2 or 3 feet, though 9 inches to a foot is more common, while the thickness varies from a half inch to 3 inches or more. The outline is usually oval, though nearly circular ones are often seen. The entire nodule is usually composed of more or less solid pyrite, often containing cavities lined with crystals, which are usually modified octahedra, rarely cubes. I have observed that these nodules often contain as a nucleus the remains of fish entirely transformed to pyrite. This was especially the case at the Hickory slope of Hazzard, Wood & Co., near Jackson Centre, Mercer county.

In the Pardoe or Brookville bed of Mercer county, the nodules are found in a zone between 1 and 2 feet above the bottom of the coal-bed, and lie with their longer axes horizontal. So abundant are they, that, though the operators are very strict in insisting that they shall all be removed from the coal in the mine, yet two men are usually employed on every tipple to pick over the coal as it comes from the screens, simply for the purpose of removing these pyrite nodules. The accompanying sections of the Pardoe bed, in Mercer county, will illustrate this occurrence of nodular pyrite.

1. *Section of Pardoe or Brookville bed, at Ormsby Shaft, near Jackson Centre, Mercer County, Pa.*

	Feet.	Inches.
1. Slate roof,		
2. Coal,	1	6
3. Bone,	0	1
4. Coal,	1	6
5. *Pyrite, nodular*,	0	0–3
6. Coal,	1	6
7. Fireclay floor,		

2. *Section of the Pardoe bed, at the Pine Grove mine, on the Shenango and Allegheny Railroad, one mile south of Grove City.*

	Feet.	Inches.
1. Slate roof,		
2. Coal.	1	0
3. Bone,	0	1
4. Coal,	2	2
5. *Pyrite, nodular*,	0	0–3
6. Coal,	1	2
7. Fireclay,		

The similarity of the two sections is obvious.

At a number of other mines in Mercer county similar sections are shown, but these are sufficient to illustrate the occurrence. In many localities, No. 5 of the above sections seems to form an almost continuous band through the coal, though always retaining its nodular character. It seems as if, at the time this member of the bed was deposited, there were centers of attraction scattered here and there through the stratum, around which as nuclei the pyrite was formed. As already mentioned, these nuclei in some cases were undoubtedly fish. It will be interesting, in this connection, to compare the section of the Pardoe bed of Mercer county with its probable representative, the "Sulphur" or "Six-foot" bed of Cambria county, bed A. A section obtained from Mr. John Fulton, of the Cambria Iron Co., shows the condition of this bed at South Fork, east of the Viaduct axis, Cambria county, Pa. This section is as follows:

	Feet.	Inches.
1. Black slate.		
2. Coal,	1	0
3. Sulphur (pyrite),	0	1
4. Coal,	1	3
5. Slate,	0	2
6. Coal,	1	0
7. Shale and sulphur (pyrite),	0	½
8. Coal,	0	8
9. Shale,		

Here No. 7 undoubtedly represents the pyrite band, No. 5. of the Mercer county section. As this bed is not worked at present in the vicinity of South Fork, I was not able to observe whether this band still retained its nodular structure; No. 3 in this section probably represents the bone, No. 3, in the Mercer county sections. Indeed, in some mines in Mercer county, this band carries a considerable amount of pyrite, but not in nodules.

2. Pyrite occurring in persistent stratified bands is nearly always more or less friable, and usually mixed with foreign matter. By far the most common occurrence in this form of deposit is in association with the so-called natural or mineral charcoal, where the pyrite occurs as a coating on the wood cell-walls, or even entirely replacing the substance of the wood. Such material as this is very difficult to separate in mining, unless the pyrite occurs in such quantity as to really replace all the woody tissue, when it becomes sufficiently firm to stand splitting off the coal without going to powder. Such an occurrence of pyrite is characteristic of bed B, the lower Kittanning, in the Clearfield and Cambria district, where a coating of mineral charcoal converted to pyrite forms a stratum on the top of the under-clay which separates the two members of the bed. This bed has been opened in numerous places along Moshannon creek, from Peale to Osceola, and along the Tyrone and Clearfield railroad from Osceola to Powelton station. A large number of these mines, however, were only worked for local consumption, and many of them are now abandoned, owing to the pyritous character of the bed. East from Munson's station on the Beech Creek railroad, the bed increases in size, and southwest from Osceola it also enlarges, so that, at Coalport on the Bell's Gap railroad, it is again a thick bed. At Munson's station and to the north and east, the coal is now being coked, while to the west at Coalport and Irvona this is the bed that is now so largely worked for coking purposes. But at either of these localities, the coal will show on examination at the base of the bed, and immediately above the under-clay, a stratum of half an inch to an inch in thickness of pyrite, which usually to the naked eye, and always on examination with a lense, shows the grain and cells of woody tissue, characteristic of the so-called mineral charcoal. This wood usually has a fragmentary and water-worn appearance, and is not found in definite stems or trunks. In some cases, however, stems resembling *Cordaites* are found mixed with the woody fragments. This layer is often overlooked by the miner, being of course left at the bottom, except where it adheres to the coal. So charac-

teristic is it, however, in this Clearfield section, that on these grounds alone the identity of the bed can be recognized with reasonable assurance, even though no rock-section showing the intervals between the coal-beds is available. Throughout this section, the Lower Kittanning is usually a double bed, and the above remarks are applicable to the upper member only, this being the one mined. In fact, the miner is generally ignorant of the existence of the lower member, though often he is separated from it by less than one foot of fire-clay. Below are given three sections of this coal-bed, showing the two members and the position of the pyrite-layer. These are selected from a number of sections, and may be taken as typical of the three localities given.

1. *Section of Lower Kittanning Coal-bed at Kyler Mine, Munson's station, Beech Creek Railroad, Clearfield County.*

	Feet.	Inches.
1. Bony and coal,	1	6
2. Coal, clean,	3	6
3. *Pyrite layer, thin,*	0	¼
4. Fire-clay,	0	10
5. Coal, "sulphury,"	1	6
6. Sandstone,		

At this mine the pyrite-layer is thin, but, as already stated, it thickens up towards Osceola (westward).

Section of Lower Kittanning at Etna Mine, near Osceola, Centre County.

	Feet.	Inches.
1. Slate and shales,		
2. Coal,	1	0
3. Bone and slate,	0	5
4. Coal,	1	11
5. *Pyrite layer, variable,*	0	¼–1
6. Fire-clay,	0	2
7. Sandstone,		

Here the lower member of the bed seems to be absent; and indeed its presence was not detected, either in the other mines in the vicinity of Osceola, or in the mines around Powelton station on the Tyrone and Clearfield railroad.

3. *Section of Lower Kittanning Bed in Columbia No. 3 Mine, at Coalport in Clearfield County.*

	Feet.	Inches.
1. Black slate,	0	5
2. Coal,	1	6
3. Bony,	0	8
4. Coal,	1	1
5. Binder slate,	0	1
6. Coal,	1	2
7. *Pyrite layer*,	0	1
8. Fire-clay,	1	6
9. Coal, lower member,	1	1

The lower member is never mined at either of these localities, being usually too full of pyrite to be marketable. While this pyrite-layer is characteristic of the Lower Kittanning in Clearfield and Centre counties, and in Cambria county along the Bell's Gap railroad, it becomes much less noticeable, or even disappears entirely, in those portions of Cambria county along the Pa. R. R. between Johnstown and Horseshoe Bend. In fact, there is a decided change in the other characters of the bed itself in this section.

Pyrite occurs to a greater or less extent in all the slate bands or partings in the coal-beds, but as a rule it is usually associated with vegetable remains, stems or roots. In some instances, however, and especially if the layer is calcareous, the pyrite is found in well-defined crystals. Examples of this occurrence may be seen in many of the mines in the Broad Top semi-bituminous region.

3. Pyrite in the form of plates or leaves through the coal may be either hard and crystalline, or soft and associated with mineral charcoal. It usually occurs in the upper portion of the coal-bed, and is local. This form cannot be separated by hand-picking. Often it is not noticeable until the coal is weathered, when it shows as a white efflorescence on the coal, due to the conversion of the sulphide to sulphate through oxidation in presence of water. A "sulphur dump" thus often looks as if it had been whitewashed, owing to the formation of ferrous sulphate and iron alums. Pyrite in this form is much more familiar to the miner as well as to the consumer of the coal, for as long as it can be easily separated in preparation the miner pays little attention to it. But, as already observed, this form cannot be separated by any methods of hand-picking. Crushing and washing will usually effect a separation, but, as the material passing into the jigs, in the usual form of washing-plant, may be of any size, from nut to the finest slime, this separation must be im-

perfect at best. When the slimes are caught in settling-tanks and utilized for coking, it is generally found that they run much higher in sulphur than the grain and nut-sizes taken direct from the jigs. This fact is recognized at some works, as, for instance, the Isabella Furnace Works at Coketon, Westmoreland county, where the ovens are charged with mixed slimes and nut-sizes, in order to equalize the amount of sulphur in the coal and make the product more constant in composition.

4. The above remarks as to the difficulty of separating this form of pyrite from the coal will apply equally well to the form next to be considered, namely, where the pyrite occurs lining cracks, joints and cavities in the coal. This mode of occurrence is always crystalline, and is evidently not synchronous with the deposition of the coal, but an after-product derived from the surrounding rocks. Whether it was deposited from solutions containing the mineral as pyrite, or derived from the decomposition of iron salts by reducing gases, or even by some reducing action of the solid constituents of the coal itself, it would be hard to say; it seems, however, to be a fact that the occurrence is quite local, and, as a rule, not in deleterious quantity. The Pittsburgh and other beds, showing strong "face" cleavage-planes and "slips," furnish good examples of this mode of occurrence. The crusts are often thinner than paper, and are seldom more than one millimeter thick. They usually show crystalline planes on the edges, as though composed of an aggregate of crystals. Sometimes they possess a radiating form, indicating that crystallization progressed outward from a central nucleus.

5. The last occurrence to be mentioned, that of the pyrite in minute rounded particles through the substance of the coal, is in reality only another form of the nodular pyrite on a microscopic scale. Its formation was evidently synchronous with the deposition of the coal. It may occur in any coal-bed, and is impossible to separate, but usually does not occur in such quantities as to injure the quality of the coal to any great extent. Many of the Pittsburgh gas-coals show this form of pyrite.

The origin of the pyrite has already been indicated in what has gone before, it now only remains to speak of this origin somewhat more fully. In the first place, where did the iron come from which furnished the basic element in the combination? Primarily from the soil in which the Carboniferous plants grew; but directly, in all

probability, from the plants themselves. The ashes of our plants, and especially of the roots, show a notable percentage of iron; and the examination of decayed wood in clay-beds, etc, always shows stains of ferric hydrate in the wood and surrounding clay. In bogs and swamps, moreover, the standing water is nearly always reddened with hydrated ferric oxide, evidently derived in large part from the decaying plants themselves. Plants, then, contain a notable amount of iron salts, which are set free during the decomposition of the decaying vegetable matter. Sulphur may have been supplied either as gypsum or other sulphate, or in the form of hydrogen sulphide. In the former case it would be required that the gypsum be decomposed in presence of the iron salts with formation of ferrous sulphate, which latter was afterwards reduced to sulphide by the decomposing organic matter; while, on the other hand, if the reducing agent was hydrogen sulphide, the pyrite would probably be formed directly. It has been demonstrated in a number of cases that the iridescent scum found on the surface of pools in marshy ground is often largely composed of pyrite, sometimes even in sufficiently large crystals to be seen with a microscope.

Whatever the mode of formation of the pyrite, the presence of some organic matter seems to have been necessary. In the case of the nodular pyrite it would seem that this was animal matter, from the presence of fish-remains in the nodules already alluded to. This, too, would account for the more solid constitution of these nodules as compared with the other forms of occurrence spoken of. The decomposable matter of a fish is more concentrated, so to speak, than that of a plant. There is more of it in a given space, and the decomposition proceeds much more rapidly. Hence the decaying fish would decompose the ferric salts much more rapidly, and cause a rapid deposition of the sulphide resulting in a definite nodule. With woody tissue the first deposition must necessarily be superficial, and then only as the solution penetrated to the body of the wood could further decomposition go on. Hence, also, the pyrite formed from such a decomposition would be of a much more friable character than that formed from the decomposition of animal matter, since it would still retain the cellular structure of the wood in which it was deposited.

SPIRALLY-WELDED TUBING.

BY J. C. BAYLES, NEW YORK CITY.

It is seldom the privilege of one who contributes to the Transactions of a technical society, to describe a new industry in which, by processes employed for the first time, are attained results of conspicuous value. There are but few such patches of virgin soil worth the tilling, within the area circumscribed by the industries which have experienced a regular and uniform development. New and startling methods in production are often announced as about to be introduced; but, as a rule, they take their place among the "large desires with most uncertain issues," or give new significance to the historic words of an eminent statesman concerning "the bright beginnings and the bitter end of a halcyon and vociferous proceeding." In this paper I shall have the pleasure of briefly describing a new industry which has been established without announcement or proclamation — the manufacture of serviceable pressure pipes of great strength, from strips of steel or iron wound spirally, heated only along the overlapping edges, welded by hammering and finished into tubes of uniform diameter, and of such length as may be found convenient in handling and transportation. As such tubes are now manufactured on a considerable scale, in a range of sizes which will soon include everything between 4 inches and 30 inches, and as much larger as may be needed, both process and product are likely to be of interest to engineers in all departments of professional work.

The raw material of the industry is the ordinary sheet-iron or steel of commerce, of such lengths and widths as it is convenient to roll. The range of the gauges of the metal which can be employed has not yet been determined. The lightest metal thus far successfully made into pipe is No. 29 iron, and the heaviest a steel gauging somewhere between 12 and 14. Heavier machines are now building which will bend steel .165 of an inch in thickness, or No. 8 of the Birmingham gauge. Probably No. 20 or No. 22 will be the lightest used in making commercial pipes; but the experiment with No. 29 was interesting as showing the readiness with which the flame can be controlled. To get a welding heat with such thin

metal and not burn or fuse it, is a delicate operation. The gauge of stock which may be used when great strength is desired depends upon the diameter of the pipe, the stiffness of the material and the rate of speed desired in running the machine. Heavier metal can be used in an 8- or 10-inch pipe than in one of 4 inches, as the greater radius or curvature involves less difficulty in bending and forming it, and its tendency to spring back after receiving the desired shape is less pronounced. However, the range of practicable gauges is great enough to meet all the requirements of strength under light or heavy pressures.

The first step in the process of manufacture is to slit the sheets into bands of the width most convenient for the production of the desired diameter of pipe. This is done with an ordinary rotary shear provided with a table and guide. The relation of width of skelp to diameter of pipe is not arbitrary. In a 6-inch pipe, for example, 6-inch, 8-inch, 10-inch or 12-inch skelp may be used. The wider the skelp, the faster the pipe is made. With 8-inch skelp 8.175 inches is added to the length of a pipe of 6 inches diameter at each revolution; with 12-inch skelp it gains 14.59 inches with each revolution. For convenience, all diameters are made from four widths of skelp, 6, 12, 18 and 24 inches.

As it is difficult to obtain sheets longer than 12 feet, it is necessary to unite the ends by lap-welding, if long pipes are desired. To make a 6-inch pipe 30 feet long from 12-inch skelp, it is necessary to have a ribbon of metal about 49 feet long. The ends of the strips of skelp are united by a machine known as a cross welder. The sheets are so placed as to give about $\frac{1}{2}$-inch lap, and in this position they are firmly clamped. Heat is then applied by furnaces above and below, which move along the seam. As they recede, the hot edges are welded between a hammer moving vertically and an anvil of reciprocal motion. To place and clamp the skelp, heat the overlapping edges and weld them, consumes about one minute to each cross seam of 12 inches. A pressure of the foot of the operator upon a treadle, engages a worm-wheel and worm which rotates a reel upon which the skelp is wound. As it is drawn from the reel, it passes between pressure-rolls which smooth out any buckling or other irregularity in the still hot metal, and rotary shears trim off the burr at the ends of the welded seam. In case the weld is defective or the sheets have not been clamped in line, the weld is cut by a shear held suspended when not in use, and the ends are welded

again. As a rule, the weld is smooth and perfect, and the extra thickness of metal at the weld occasions no inconvenience in forming the pipe.

The pipe-machine is quite simple in construction, and is chiefly made of heavy castings, requiring but little finish. It occupies about 3 x 6 feet of floor-space.

The reel carrying the ribbon of skelp is put in position and one end of the metal is placed upon the guide-table, which is set at the angle due to the width of the skelp and the diameter of the pipe into which it is to be made. The metal is carried into the machine between feed-rolls geared together, which are actuated by a ratchet, giving them an intermittent rotation, and a rate of feed variable between $\frac{1}{20}$ and $\frac{1}{8}$ of an inch at each impulse, at the pleasure of the operator. This carries it into the forming-jaws, which bend it to the desired curvature—the forming being effected by pinching the metal in curved jaws.

The essential features of the pipe-machine are a guide-table for the skelp, adjustable to the desired angle; feed-rolls, to pass it forward with an intermittent progress, so that it shall advance when the hammer is raised and be at rest when the hammer falls; a former, to curve the metal to the desired radius, also adjustable; a furnace, to heat the metal; a hammer, to weld it, and an anvil to support the pipe and receive the shocks of the hammer. No mandrel is used. The pipe in the forming process is held in place by a pipe-mold, which is a cylindrical shell, within which the pipe rotates as the stock is fed in. The anvil is of considerable mass, steel-faced, and extends the entire width of the skelp. The hammer is light, and at normal speed strikes 160 blows per minute. The heating is done in a furnace so constructed as to heat both the edges to be united for the space of several inches ahead of the point at which the welding is effected. The upper skelp enters the furnace flat, and the lower skelp curved, having already been through the forming-jaws. The heat is imparted by one or two blow-pipes of water-gas and air, discharging upon the metal through passages of suitable form in the refractory lining of the furnace. One gas-flame has been found sufficient, but two work better; and besides being more convenient to control, they heat the metal more rapidly and permit an accelerated feed. As very little gas is wasted, the greatest economy attends the most rapid production of pipe, irrespective of the quantity burned, which in any case is about 30 feet per foot of

welded seam. The speed of production depends, as stated, upon the thickness of stock to be heated, and the relation of width of skelp to diameter. It averages a foot per minute to each machine, and it is probable this average can be raised considerably. The machines are so nearly automatic in operation that very little skilled labor is needed in running the plant. The operator has his gas, air and feed under control by convenient means, and varies their relations until he has them just as he wants them. He can see the edges as they emerge from the furnace, and about all the skill he needs is that which will enable him to judge by its color whether the iron is above, below or at the welding-heat. Unskilled labor prepares the stock and removes the finished product. The ends of the pipe are cut square by suitable machinery without reversing, and after testing and treating with asphalt, the pipe is ready for shipment. As may be supposed, all the difficulties of mechanical development have centered in the pipe-machine. To make this satisfactory in operation has been no more serious task than is usually entailed in the effort to make old mechanical motions perform new functions. It has been accomplished within eighteen months; and if the machines for the several sizes completed are not yet perfect, they have at least reached the condition of practical excellence in operation which gives assurance that they can be multiplied with profit. Six have already been built, with a range in diameters of from 4 to 16 inches, and others are in progress which, as before stated, will extend the range to 30 inches.

Spirally-welded pipe does not depend upon new uses for its introduction, but being entirely practical for all purposes, enters the market as a competitor with lap-welded pipe. The advantages inherent in a pipe formed in winding a blank spirally, are obvious. All tests of rolled iron show that it is materially stronger with the "grain" than across it. Fairbairn's well-known experiments prove that the difference is fully 10 per cent. on the average. Skelp-iron, of which all forms of wrought-iron pipe are made, must of necessity be slit into bands in the direction of its fibers. If the pipe is formed by curving the strip of skelp over a mandrel of any sort, and finishing it by a longitudinal "lap"- or "butt"-weld, an inherent structural weakness is inevitable. A bursting-strain, exerted circumferentially, brings upon the iron a stress which it is much less able to resist than it would be if the fibres and weld ran the other way. A bursted wrought-iron pipe, butt-welded or lap-welded, is always

found to be split lengthwise, usually along the line of the weld, which is never as strong as the body of the stock from which the pipe is made. It is usually possible, with a wooden plug driven into the end, to split such a longitudinally-welded pipe along the line of the weld, without serious difficulty. In a pipe formed spirally, the bursting-strain is exerted largely in the direction in which the fibers of the iron run; and the spiral weld, instead of being a line of structural weakness, is a reinforcement, giving the pipe added stiffness, and making it, when finished, stronger to resist bursting- or crushing-strains than the stock of which it is made would be in any other form. This kind of pipe permits the use of very much lighter stock than can be employed in other forms, and gives great strength with gauges of iron which, if heated to the welding-temperature prior to forming, would collapse and defeat all efforts to form and weld them.

When steel is used, the advantages incident to spirally winding are less conspicuous than in the case of iron, but owing to the greater strength of the material it can be used of so much lighter gauge that a very important economy is secured, both in first cost and in transportation and handling. Concerning the spiral weld it is enough to say it is a perfect weld. Every known test has been applied to it, and the results have been in every instance fracture in the body of the stock and not in the weld.

The strengths attainable in light pipe, if the material is used to the best advantage, are quite surprising. A 6-inch pipe made of No. 14 gauge iron of good average quality, showing under test 33,000 pounds elastic limit, and 50,000 pounds ultimate strength, has a proof strength of 913 pounds per square inch, and an ultimate strength of 1383 pounds per square inch. A 12-inch pipe of the same stock has a proof strength of 456 pounds, and an ultimate strength of 691 pounds. If a good grade of soft steel is used instead of iron, the 6-inch pipe will carry 1106 pounds pressure without deformation, and will not burst under 1800 pounds; the 12-inch pipe can be tested to 475 pounds and will carry 900 pounds without fracture. This is very practical pipe. Using the same diameters and gauges of stock for comparison, we find that the 6-inch spirally welded pipe weighs 5.2 pounds per foot against 18.77 pounds per foot for standard lap-welded pipe, and 28.28 pounds for medium cast-iron pipe: the 12-inch spirally welded pipe weighs 10.46 pounds against 54.65 pounds for lap-welded and 77.36 for medium cast-iron. I give below tables of strength and weights:

SPIRALLY-WELDED STEEL TUBES.

Proof strength of stock 40,000 lbs. per square inch.
Ultimate " " " 65,000 " " "

Diameter of Pipe.	Gauge.	Proof Strength per Sq. In.	Ultimate Strength per Sq. In.
4	18	980	1593
	16	1300	2113
	14	1660	2697
5	18	784	1274
	16	1040	1690
	14	1328	2158
6	18	653	1062
	16	866	1408
	14	1106	1798
8	18	490	796
	16	650	845
	14	830	1079
	12	1090	1771
	10	1340	2177
	8	1650	2681
10	18	392	637
	16	520	845
	14	664	1079
	12	872	1417
	10	1072	1742
	8	1320	2145
12	18	326	531
	16	433	704
	14	474	899
	12	727	1181
	10	893	1451
	8	1100	1788
14	18	280	455
	16	371	603
	14	474	771
	12	623	1012
	10	765	1244
	8	943	1532
16	18	245	398
	16	325	528
	14	415	674
	12	545	886
	10	670	1088
	8	825	1341
18	18	218	354
	16	289	469
	14	369	599
	12	484	787
	10	596	968
	8	733	1192
20	18	196	319
	16	260	422
	14	332	539
	12	436	709
	10	536	871
	8	660	1073
22	18	178	289
	16	236	384
	14	302	490
	12	396	644
	10	487	791
	8	600	975
24	18	163	265
	16	216	352
	14	277	449
	12	363	590
	10	446	726
	8	550	894

SPIRALLY-WELDED IRON TUBES.

Proof strength of stock 33,000 lbs. per square inch.
Ultimate " " " 50,000 " " "

Diameter of Pipe.	Gauge.	Proof Strength per Sq. In.	Ultimate Strength per Sq. In.
4	18	809	1225
	16	1072	1625
	14	1369	2075
5	18	647	980
	16	858	1300
	14	1096	1660
6	18	539	816
	16	715	1083
	14	913	1383
8	18	404	613
	16	536	813
	14	684	1037
	12	899	1362
	10	1105	1675
	8	1361	2062
10	18	323	490
	16	429	650
	14	548	830
	12	719	1090
	10	884	1340
	8	1089	1650
12	18	269	408
	16	357	544
	14	456	691
	12	599	908
	10	737	1117
	8	907	1375
14	18	231	350
	16	306	464
	14	391	593
	12	512	779
	10	632	967
	8	778	1179
16	18	202	306
	16	263	406
	14	342	518
	12	449	681
	10	552	837
	8	680	1031
18	18	179	272
	16	238	361
	14	304	461
	12	399	605
	10	491	744
	8	605	917
20	18	162	245
	16	214	325
	14	274	415
	12	360	545
	10	442	670
	8	544	825
22	18	147	221
	16	195	296
	14	249	377
	12	327	495
	10	402	609
	8	495	750
24	18	135	204
	16	179	271
	14	228	346
	12	299	454
	10	369	558
	8	454	687

COMPARATIVE WEIGHTS OF PIPES, PER LINEAR FOOT, WITHOUT COUPLINGS OR HUBS.

SPIRALLY-WELDED STEEL OR IRON.							LAP-WELDED IRON.	CAST-IRON.		
Diam. of Pipe.	No. 18.	No. 16.	No. 14.	No. 12.	No. 10.	No. 8.	Standard.	⅜ In. Thick.	⅝ In. Thick.	¾ In. Thick.
in.	lbs.	lbs.	lbs.	lbs.	lbs.	lbs.				
4	2.06	2.73					10.72	22.05	28.28	34.94
5	2.58	3 42					14.56	26.94	34.34	42 38
6	3.09	4.10	5 20				18.77	31.82	40.56	49.6
8	4.13	5.47	6.98				28.35	41.64	52.68	64.27
10	5.16	6.83	8.72	11.44	14.08		40.64	51.46	65.08	78.99
12	6 18	8.20	10.46	13.74	16.90	20.80	54.65	61.26	77.36	93.7
14	7.22	9.57	12.20	16.02	19.72	24.86		71.07	89.61	108.46
16	8.25	10.93	13.95	18.30	22.54	27.72		80 87	101.82	123.14
								⅝ In. Thick.	¾ In. Thick.	⅞ In. Thick.
18	9.28	12.30	15.69	20.69	25.35	31.19		114.1	137.84	161.9
20		13.66	17.44	22.28	28.16	34.66		126.33	152.53	179.02
22		15.03	19.18	25.27	30.98	38.12		138.6	167.24	196.46
24		16.40	20.92	27.48	33.80	41.60		150.85	181.92	213.28

The question of durability in service is one which naturally suggests itself when light steel or iron pipes are discussed. Experience on the Pacific Coast seems to have settled this question, as the cheap expedients adopted for water-conveyance during the days when hydraulic mining was most extensively conducted have been followed ever since in permanent engineering works. The best attainable data on this subject which I have found, are presented in a paper read by Hamilton Smith, Jr., before the British Iron and Steel Institute, and printed in vol. i. of the *Journal* for 1886. Much of the information contained in this paper is quite surprising, especially in the case of the two mains across Humbug Cañon. These pipes were laid in 1868. They are of 26 inches diameter, 1194 feet long, of common iron $\frac{1}{18}$ inch in thickness, single riveted. During all this time they have been delivering water under 120 feet head, and Mr. Smith gives the maximum tensile strain in pounds on the metal per square inch as 11,500. Large as these figures look, they are simply the result of applying to the conditions given Rankine's well-known formula for their cylindrical shells.

Riveted pipe in its best estate labors under the disadvantage of inherent structural weakness, and liability to rust between the over-

lapping edges and around the rivets. Pipes of this character on the Pacific Coast are very roughly tarred in position, and the coating is quite liable to be pulled off by the adobe clay in which most of them are laid; but they have a record of useful life since 1853, and many towns are supplied with water under considerable heads from pipes of this kind which have been more than 20 years in service. A welded pipe carefully coated with asphalt should, with fair treatment, have a record at least as good, and probably much better.

The coupling of light pressure-pipes involves no difficulties, but it entails new methods. These are convenient and inexpensive, and make perfectly tight joints. The couplings are chiefly of cast-iron, and their form depends upon the service in which the pipe is to be employed. Steam, water, petroleum, compressed air and gas, all present different problems in couplings, but no difficulties which have not already been fully met. The couplings are as practical as the pipe.

The field of pre-eminent and unique advantage which belongs to spiral-welded pipe is in the large sizes. The present practical diameters in lap-welded pipe are 20 inches and under; the lengths are dependent upon the size of the heating-furnaces. Spiral-welded pipe can be made of any diameter required, and of any length which can be handled in transportation. Its uses are too varied for enumeration. They include all the operations of hydraulic mining, mine-draining and ventilation, the conveyance and distribution of water, natural gas, illuminating gas, steam, compressed air and petroleum. There is a large and unsatisfied demand for sizes of wrought-iron and steel pipe for uses in which cast-iron is too costly or cannot be trusted, which seems to be fully met by the spirally-welded tubing. Perhaps the most interesting fact in connection with its manufacture is the number of new and unexpected uses opening for it, in short lengths, from 12 inches to 4 feet. These seem great enough in the aggregate to absorb the production of more machines than one mill of convenient proportions will hold; and they have a special interest for the manufacturers, in that they seem to obviate all danger of waste from accidents resulting in defects disclosed in the testing of long tubes. Experience contradicts prophecy in this respect, and it is found that the problem is not how to make long pipes, but how to cut long pipes into short pieces to the best advantage.

THE BOFORS STEEL CAST GUNS.

BY O. E. MICHAELIS, PH.D., CAPTAIN OF ORDNANCE, U. S. ARMY,
AUGUSTA, ME.

(Boston Meeting, February, 1888.)

I AM indebted to Professor Richard Åkerman, of the Stockholm School of Mines, a high authority on steel, for the data, a translation of which is herewith offered to those interested. The professor informs me that a 6-inch gun has been cast since the tests described were undertaken.

The Swedish authorities consider the Bofors guns as reliable as those heretofore purchased from Armstrong and Krupp. Certainly a careful consideration of the sterling physical qualities developed by the Bofors steel, and their subsequent confirmation in actual, exhaustive trial, must greatly encourage all who believe in the thorough investigation of the feasibility of our casting steel guns. It will be urged that the pieces submitted to such crucial tests were of small caliber: true; but Rodman began with 8-inch, continued with 10- and 15-inch, astonished the world with the huge 20-inch, and had actually designed a 30-inch thunderer at the close of our war.

The first pneumatic dynamite gun was a two-inch tube, but under the enthusiastic superintendence of Captain Zalinski it has gradually grown to 15 inches, and has aroused the world's interest.

Progress is a synonym of development; the experience gained with small calibers will lead to their increase. Such must be the result, unless it can be shown that there is an intrinsic narrow limit which cannot be passed. I am not of those who believe in the existence of a steel cast "so far but no farther."

The symmetry of the Swedish metric data disappears in the translation.

MANUFACTURE.

The metal used at the Bofors Works in gun-manufacture is unforged open-hearth steel, cast without blow-holes.

It has from time to time been asserted that it is impossible to pro-

duce open-hearth steel with sufficient constancy to warrant its use as a gun-metal, possessing all required guarantees resulting from homogeneity. Although the Martin-Siemens process is the only one that affords the utmost facility for testing with precision at every period of fusion the condition of the steel, yet, generally speaking, the criticism is founded in truth. The fact is that, especially in the fabrication of open-hearth steel, ingredients are used whose quality leaves much to be desired. And, as a rule, this must be the case; for the Martin process presents this advantage, from the standpoint of economy, that, with the exception of manganiferous pig, forming about 30 per cent. of the bath, the rest may be made up of unselected scrap of every description. And, further, the pig itself is not too frequently of satisfactory brand. Even when the best ores are used for the production of the pig, charcoal is rarely used in the blast-furnaces, outside of Sweden. For this fuel, comparatively costly, coke is usually substituted, less expensive, but which as an offset produces a much less pure pig. This does not obtain at Bofors. Not only are ordinary pig and spiegeleisen made from the very best Swedish ores, but also the only fuel employed is charcoal. In addition, the iron used for decarburization is made at Bofors, specially for the purpose, of Lancashire ores, generally acknowledged as the best in the market. We must add that wood alone is used in the generators.

Under these conditions the same excellent product is always secured. As already stated, at every moment and with absolute accuracy, the carbon-point of the bath may be determined, by the examination of small specimens taken from the furnace. It is clear that a more homogeneous metal can hardly be obtained by any other process.

We cannot deny the good effect of forging upon honeycombed steel blocks, for it is the only means of rendering such steel fit for technical purposes; but on the other hand, it must be admitted that it is both legitimate and useful to endeavor to make steel which shall be entirely free from blow-holes, without the necessity of subjecting it to hammering. Indeed the forging of great blocks of steel is too delicate an operation to be always successful; and more, it is extremely costly. Before the block is put under the steam-hammer, it must be brought to the proper temperature; there is risk of over- or under-heating. The hammering, too, must stop at the right moment, for the block may be fissured if it be hammered after becoming too cool. Owing to these circumstances, and also to others, it occasionally occurs that the blocks are so injured under the hammer as to

necessitate their relegation to the scrap-heap. Sometimes these injuries are not made manifest until the machining of the block is well advanced; sometimes they remain altogether undiscovered. As a matter of fact, at least when it is a question of heavy guns of forged steel, it is necessary to make use of blocks that have numerous and perfectly apparent defects, for if this were not done, the rejections would be too many, and would greatly enhance the cost of the undertaking.

To avoid the difficulties and the risks inherent in the hammering of huge blocks of steel, the Bofors Works, as at Terre Noire, avails itself of the method of casting steel without blow-holes. As it is well known, this method consists in the introduction of ferro-silicon and ferro-manganese in proper quantity just before drawing the bath. Unquestionably the steel obtained by this process is free from blow-holes, if the fusion has been well managed; but the fracture is crystalline and very bright, and, though this comparatively crude product shows great elasticity and tenacity, it is yet far from possessing completely these two qualities, and especially ductility, to a sufficiently high degree, although it acquires them at once by the succeeding treatment.

This treatment is as follows: The cast block is thoroughly reheated and then buried in pulverized cinders, where it remains until completely cooled. As a rule, after this operation, the block possesses very great elasticity and tenacity, and at the same time, very considerable ductility. After being rough-finished and bored to a diameter slightly less than the regulation caliber, the block is again reheated, and oil-tempered. It is then annealed at a very low temperature to counteract the tensions engendered by the oil-tempering. The block is now ready for finishing. If the carbon-point lies between 0.30 and 0.40, the bars, cut from the block, show all the physical qualities which characterize the best crucible steel of mean hardness. The fracture is also fine in grain and of silken sheen, appropriate for this steel. We append the results of tension-tests of the steel used in making various experimental Swedish army and navy guns.

The density of the steel is 7.82–7.84.

Firing-Test.

The first experimental piece was a 4-pounder smooth bore ordered by the navy. This caliber was selected owing to the fact that in

Sweden 4-pounder test-tubes are always cast at the same time as are the lining cylinders of heavy cast-iron steel-banded guns. This small piece is submitted later to endurance-tests, if the inspecting officer deem it necessary, according to a programme established once for all. Consequently a very complete knowledge is thus gained of the strength of the guns; and it was deemed desirable to obtain comparative results for the old and new metal.

Construction-Data of the Four-Pounder Smooth Bore.

Caliber,	3.295 inches.
Total length,	59.056 "
Length of bore,	57.441 "
Maximum exterior diameter, over powder-chamber,	11.164 "
Diameter of powder-chamber,	3.295 "
Weight of gun,	2445 lbs.

Before finishing, the gun was annealed in pulverized cinders, but was not oil-tempered. The physical test to which a specimen taken from the sprue-head, the worst end of the block, was submitted, gave the following results:

Elastic limit (in pounds per square inch),	31,700
Tenacity " "	87,300
Coefficient of elasticity,	33,000,000
Elongation in 3.937 inches (100 millimeters), per cent.,	13
Contraction in per cent. of original section,	14.4

The carbon-point was 0.37. The following table exhibits the results of the endurance-test:

4-POUNDER SMOOTH-BORE.

Number of Round.	Number in Each Series.	Weight of Charge.	Weight of Shot.	Maximum Pressure.	Maximum Enlargement of Chamber.	
					Horizontal.	Vertical.
		Pounds.	Pounds.	Pounds.	Inches.	Inches.
1	1	1.65	11.02	.....	.....	.....
2-3	2	"	8.82	.....	.....	.....
4-8	5	"	13.23	.....	.....	.....
9-13	5	2.20	"	.....	.....	.....
14-18	5	2.49	"	.....	.....	.....
19-23	5	2.76	"	.....	.....	.....
24-108	85	"	17.64	26000	0.000	0.000
109-113	5	"	22.05	.....	.....	.....
114-118	5	"	26.46	.....	.....	.....
119-123	5	"	30.87	.....	.....	.....
124-128	5	"	35.28	.....	.....	.....
129-133	5	"	39.68	.....	.....	.....
134-138	5	"	44.10	.....	.....	.....
139-143	5	"	48.50	.....	0.000	0.000
144-148	5	"	52.91	.....	.....	.....
149-153	5	"	57.32	27500	.....	.....
154-178	25	3.38	"	33800	0.002	0.002
179-188	10	3.86	"	36400	.....	.....
189-198	10	4.41	"	41400	0.004	0.004
199-208	10	4.96	"	41700	.....	.....
209-218	10	5.51	"	54000	.....	.....
219-300	82	6.06	"	69000	0.006	0.008
301	1	2.76	"	28000	.....	.....
302	1	3.38	"	36300	.....	.....
303	1	3.86	"	38600	.....	.....
304	1	4.41	"	36100	.....	.....
305	1	4.96	"	47500	.....	.....
306-308	3	5.51	"	62400	0.007	0.008
309-311	3	6.06	"	55000	0.007	0.008
312	1	2.76	.....	33900	.....	.....
313-314	2	3.38	.....	46000	.....	.....
315	1	3.86	.....	44500	.....	.....
316	1	4.41	.....	57700	.....	.....
317	1	4.96	.....	51600	.....	.....
318	1	5.51	.....	55500	0.017	0.018
319-322	4	6.06	.....	70000	0.019	0.022

Total number of rounds fired—322.

It should be stated that, up to the 144th round, straw had been used between the last projectile and the charge, and thin wooden disks between the projectiles, which were solid and of cylindrical form. As no extraordinary enlargement had occurred during the first 300 rounds (see Table), musket-powder was substituted at this

stage for the small-grained cannon-powder heretofore used. However, the pressure was not increased—easily explained by the fact that considerable erosion had occurred at the seat of the projectile, an unavoidable effect, due to the necessity of using windage-projectiles in this smooth-bore gun.

From the 312th round cannon-powder was used, and at the same time the method of testing was entirely changed. After having inserted the charge, the bore was filled with solid plugs and the gun laid horizontally, so that the forward plug, projecting from the muzzle, rested against a cast-iron block weighing over 2500 lbs. and firmly fixed in position. The gun thus laid was then fired. Usually the gun, in recoiling, was received in a sand rampart made for the purpose. Sometimes, however, it flew up, presenting the strange sight of a cannon gyrating among the tree tops. At the 11th round, with the cannon itself acting as a projectile, that is to say, after 322 rounds of proof-firing, the gun burst into several pieces.

The strongest cast-iron guns of the same construction tried to destruction had endured only two hundred and nineteen proof-rounds. The exceedingly favorable result of the first trial of the new gun-metal induced the Swedish Ordnance authorities to experiment with a 4.6 inch steel cast siege-gun.

Construction-Data of the 4.6-inch Gun No. 1.

Caliber,	inches,	4.60
Total length (29.9 calibers),	"	137.80
Length of bore (28.5 calibers),	"	131.40
Exterior diameter, maximum, over powder-chamber,	"	12.60
Exterior diameter of body, over powder-chamber,	"	7.83
Diameter of chamber,	"	4.88
Number of grooves,		36
Depth of grooves,	inches,	0.049
Width of lands,	"	0.118
Final twist, in calibers for one turn,		30
Cross-section of bore,	square inches,	17.13
Volume of bore,	cubic inches,	2282
Capacity of chamber (shell in place),	"	267
Weight of gun,	pounds,	2628

Otherwise the construction of the piece is very much like the Krupp mantle-guns, except that it has the screw, instead of the wedge, fermature.

The two main parts of the gun, to wit, the mantle and the body, as well as the breech-screw, were annealed in pulverized cinders before machining, but were not tempered.

The tension-tests, of bars taken from the best ends of the two parts, gave the following results:

	Mantle.	Body.
Carbon-point, ...	30	25
Elastic limit, ...	33,500	42,600
Tenacity, ...	77,600	81,800
Coefficient of elasticity	32,300,000	32,300,000
Elongation in 3.937 inches (100 millimeters),	23.5 per ct.	21.5 per ct.
Reduction of area at rupture, . . .	30.8 per ct.	21.9 per ct.

The endurance-test is shown in the following table:

4.6-INCH GUN, No. 1.

Number of Round.	Number of Rounds in Each Series.	Kind of Powder.	Weight of Charge.	Capacity of Chamber per lb. of Charge.	Weight of Projectile.	Initial Velocity.	Energy of Shot.		Maximum Pressure.	Maximum Enlargement of Chamber.	
							Total.	Per pound of Weight of Gun.		Horizontal.	Vertical.
			lbs.	cu.in	lbs.	ft.	ft-tons.	ft-lbs.	lbs.	in.	in.
1-2	2	$\frac{2}{10}$ gr. 1877.	7.50	78.7	33.7	1621	613	1157	27800	.....	.....
3-17	15	"	7.72	74.5	33.7	1631	623	1172	28400	.....	.....
18-32	15	"	7.72	74.5	36.8	1568	630	1179	33400	.....	.....
33-34	2	$\frac{2}{10}$ gr. 1880.	7.72	74.5	33.7	1667	649	1215	39800	.....	.....
35-47	13	"	7.39	79.3	33.7	1624	617	1157	31300	0.002	0.001
48-173	126	$\frac{2}{10}$ gr. 1877.	7.72	76.9	36.8	.....	.....	.....	.....	0.005	0.004
174-400	227	"	7.72	76.9	36.8	.....	.....	.....	.....	0.005	0.004
401-600	200	"	7.72	76.9	36.8	.....	.....	.....	.....	0.007	0.007
601-800	200	"	7.72	76.9	36.8	.....	.....	.....	.....	0.010	0.010
801-1000	200	"	7.72	76.9	36.8	.....	.....	.....	.....	0.011	0.009
1001-1006	6	"	8.27	70.8	36.8	1627	675	1303	34400	.....	.....
1007-1016	10	"	8.82	66.5	36.8	1726	762	1432	43600	.....	.....
1017-1020	4	$\frac{2}{10}$ gr. 1880.	8.27	70.8	36.8	1670	713	1338	47300	0.011	0.010
1021-1040	20	"	7.50	78.7	33.7	.....	.....	.....	.....	.....	.....
1041	1	$\frac{2}{10}$ gr. 1877.	8.82	65.9	73.6	.....	.....	.....	80800	0.069	0.075

Total number of Rounds fired—1041.

During these tests Swedish "large grain" cannon-powder was always used. Although the grains of this powder are relatively small, it is, however, a little less violent than that of .25 to .40 inch granulation (Hamm or Rottweil), as it was not milled under stone. Still, the two-tenth powder is yet somewhat too quick for a 4.6 inch

gun. The maximum powder pressures show, consequently, much variation, dependent upon the year of manufacture. Notwithstanding this drawback, the gun has stood well these severe tests.

In accordance with the programme, the gun was first fired one thousand regulation rounds. This firing produced no erosions whatever; the diameter of the chamber was enlarged about 0.01 inch, due to the great malleability of the metal.

It was noted that after the 800th round there was no augmentation of the enlargement. The extremity-tests began with round No. 1001, but were interrupted for a special trial (rounds 1021–1040). It should be noted that after the rounds with heavy charges (Nos. 1001–1020), during which the pressures rose to 34,400–47,800 pounds, no increase of enlargement was observed. There was no erosion of the bore, and no scoring was evident in front of the cone joining chamber and bore.

When the extremity-test was resumed, the gun burst at the first round (charge 8.82 lbs., projectile 73.6 lbs.). In this round the gas-pressure unexpectedly rose to 80,800 lbs.

The cause of this abnormal pressure was clear when the projectile next to the charge was recovered. In order to use the projectiles over again, they had been made of Swedish gun-iron of great tensile strength, but as a result exceedingly soft. The projectile was consequently upset under the pressure of the gas, and was slugged through the bore as a lead ball would be. Indeed the lands cut into the base of the projectile to the depth of 0.04 inch. The gun burst in this way: the part of the mantle behind the front thread of the breech-screw seat was torn off and broken into two large pieces, which, together with the breech-screw and gas check-ring were projected to the rear. The body of the gun, on the other hand, remained completely intact.

There were no signs of cracks, although the chamber had been enlarged in one round to the very large amount of 0.06 inch. It must further be added, in connection with the great strength of the gun-body, that the maximum exterior diameter over the chamber of this piece, at once long and light, was only 12.6 inches, a comparatively small dimension (the corresponding diameter of the light Krupp 12 per cent. gun is 13.4 inches). The trials having been thus, in general, very successful, the Swedish Ordnance authorities determined to continue them upon a larger scale, and ordered for this purpose four guns, three 3.3-inch field-guns and one 4.6-inch siege-

piece. As the steel of the first 4.6-inch gun had shown itself a trifle soft, it was determined that the carbon-point should be slightly raised, and that the blocks should be oil-tempered.

Construction-Data of the 3.3-*inch Guns and the* 4.6-*inch Gun No.* 2.

	3.3-Inch Field-Gun.	4.6-Inch Siege-Gun.
Caliber.....inches	3.3	4.6
Total length..... "	90.5	137.8
Total length.....calibers	27.4	29.9
Maximum exterior diameter over chamber, inches	9.05	12.76
Diameter of gun body, over chamber..... "	5.91	7.83
Diameter of chamber..... "	3.94	4.80
Number of grooves.....	24	36
Depth of rifling.....inches	0.039	0.049
Width of lands..... "	0.138	0.118
Final twist, in calibres for one turn.....	33	30
Cross-section of bore.....sq. inches	8.59	17.13
Volume of bore.....cu. inches	751	2282
Capacity of chamber (shell in place)..... " "	109	267
Weight of gun.....pounds	1010	2731

In construction, the 3.3-inch guns are exactly like the field-guns furnished the Swedish artillery by Krupp—mantle-guns with cylindro-prismatic wedge. The 4.6-inch gun No. 2 was constructed very much like No. 1. But as the seating of the Broadwell ring decidedly weakened the end of the gun-body, made apparent by the enlargement of the back-end of the chamber, an enlargement which diminished toward the front, the new piece was reinforced, after the Krupp method, by a band placed centrally over the ring-seat.

Before machining, the mantle and body-blocks were annealed in pulverized cinders.

To make the metal harder, and to enable it to better resist permanent deformation, the various parts, as has already been mentioned, were oil-tempered after rough finishing and first boring.

The results of tension tests of bars cut from the best ends of the various parts are given below:

Tension-Tests.

Designation of Gun.	Name of Part.	Carbon Point.	Elastic Limit.	Tenacity.	Coefficient of Elasticity.	Elongation in 3.937 in.	Reduction of Area at Rupture.
			Lbs. per sq. in.	Lbs. per sq. in.		Per cent.	Per cent.
3.3-in. No. 1.	Mantle.	.30	48000	85900	35100000	21.0	48.9
"	Body.	.30	49000	89500	36700000	20.0	49.0
3.3-in. No. 2.	Mantle.	.35	53700	84800	33400000	21.2	42.3
"	Body.	.37	70400	111900	32600000	14.8	36.3
3.3-in. No. 3.	Mantle.	.38	53700	100300	32700000	19.0	33.5
"	Body,	.38	73500	121500	34600000	9.0	22.1
4.6-in. No. 2.	Mantle.	.39	55900	95000	33100000	14.0	27.8
"	Body.	.35	33600	85000	38000000	15.0	19.0

NOTE.—8.3-in. Gun, No. 1, was cast before it was determined to raise the carbon-point.

As the elastic limit and the tenacity for gun-body No. 3 far exceeded the required limit, while on the other hand its ductility was lower than usual, it was thought best to soften it by a special operation. This undoubtedly beneficial procedure was, however, to avoid loss of time, not undertaken.

Bars taken from the worst ends of the castings were, as a rule, weaker than those taken from the best ends. The least favorable result from the worst bars was about the same as for gun No. 3. It had an elastic limit of 38,400 lbs., a tenacity of 86,300, and a stretch in 3.937 inches of 3.4 per cent. The end here called the worst is sometimes as good as, even though rarely better than, the other. It must further be stated that the variation between the two ends of the parts has little effect upon the strength of the guns, because it is very small, and because the best end is always toward the breech.

The four tables following give the result of the practice.

Maximum Pressure.	Maximum Enlargement of Chamber.	
	Horizontal.	Vertical.
lbs.	in.	in.
27600	0.000	0.000
31800	0.000	0.000
33600	0.000	0.000
35400	0.000	0.000
43000	0.004	0.002
43900	.....	.....
39400	.....	.....
43800	.....	.....
46900	.....	.....
48600	0.056	0.028
45000	.....	.....
56600	0.083	0.050
50800	.....	.....
62100	0.108	0.074
72700	.....	.....

Maximum Pressure.	Maximum Enlargement of Chamber.	
	Horizontal.	Vertical.
lbs.	in.	in.
27600	0.000	0.000
34100	0.000	0.000
38600	0.000	0.000
38900	0.000	0.000
43200	0.000	0.000
51000	.....	.....
41300	.....	.....
49400	.....	.....
55800	.....	.....
56000	0.045	0.022
54700	.....	.....
68200	0.057	0.033
64500	.....	.....
59200	0.060	0.037
65100	.....	.....
75400	0.171	0.138

3.3-INCH GUN No. 3.

Number of Round.	Number of Rounds in Each Series.	Kind of Powder.	Weight of Charge.	Capacity of Chamber per pound of Charge.	Weight of Projectile.	Initial Velocity.	Energy of Shot.		Maximum Pressure.	Maximum Enlargement of Chamber.	
							Total.	Per Pound of Weight of Gun.		Horizontal.	Vertical.
			lbs.	cu.in	lbs.	feet.	ft.-tons.	ft.-lbs.	lbs.	in.	in.
1-2090	2090	$\frac{5}{16}$ gr. 1880.	3.38	72.6	14.8	1555	247	1208	27600	0.000	0.000

Total number of rounds fired—2090.

4.6-INCH GUN, No. 2.

Number of Round.	Number of Rounds in Each Series.	Kind of Powder.	Weight of Charge.	Capacity of Chamber per pound of Charge.	Weight of Projectile.	Initial Velocity.	Energy of Shot.		Maximum Pressure.	Maximum Enlargement of Chamber.	
							Total.	Per Pound of Weight of Gun.		Horizontal.	Vertical.
			lbs.	cu.in	lbs.	feet.	ft.-tons.	ft.-lbs.	lbs.	in.	in.
1-7	7	$\frac{5}{16}$ gr. 1880.	7.17	81.2	37.2	1535	605	1121	32000	0.002	0.002
8-14	7	$\frac{5}{16}$ gr. 1881.	7.17	81.2	37.2	1542	610	1135	31600	.....	.....
15-17	3	$\frac{5}{16}$ gr. 1882.	7.17	81.2	37.2	1526	597	1107	29000	.....	.....
18-20	3	"	7.39	78.7	37.2	1565	630	1172	38800	.....	.....
21-25	5	$\frac{5}{16}$ gr. 1877.	7.39	78.7	36.8	.....	.....	.....	29700	.....	.....
26,27	2	"	7.72	75.1	36.8	.....	.....	.....	42800	.....	.....
28-150	123	"	7.39	79.3	36.8	.....	.....	.....	.....	0.011	0.013
151-536	386	$\frac{5}{16}$ gr. 1882.	7.39	79.3	37.0	1558	623	1157	27700	.....	.....
537-618	82	"	3.53	166.6	37.0	.....	.....	.....	.....	.....	.....
619-755	137	"	1.43	410.7	37.0	.....	.....	.....	.....	0.018	0.019
756-989	234	$\frac{5}{16}$ gr. 1881.	7.39	79.3	36.8	.....	.....	.....	.....	0.018	0.016
990-1189	200	$\frac{5}{16}$ gr. 1877.	7.39	79.3	36.2	.....	.....	.....	.....	0.019	0.020
1190-1289	100	$\frac{5}{16}$ gr. 1881.	7.39	79.3	36.8	.....	.....	.....	.....	0.018	0.020

Total number of rounds fired—1289.

According to the programme of firing, each of the three 3.3-inch guns were to be fired two thousand rounds with standard charge and projectile, and thereafter two of them were to be subjected to extreme proof. The 4.6-inch gun was to be fired one thousand regulation rounds, and then to be subjected to extreme proof.

This programme was carried out without change, except that the extreme proof of the 4.6-inch has not yet taken place. After the two thousand rounds with standard charge and projectile there

were no signs of scoring in any of the 3.3-inch guns, and no discoverable enlargement of chamber. After the 2060th round with gun No. 1, and the 2080th with No. 2, the enlargement could no longer be observed. Generally speaking, the enlargement, after the same number of rounds, is greater in the gun of softer and weaker steel (No. 1) than in the other. It is worth special notice that the horizontal enlargement greatly exceeds the vertical. The reason of this is that the breech-slot, which is very large and traverses the mantle from side to side, weakens the gun considerably in a horizontal direction.

Gun No. 1 burst at the 2152d round into several pieces. On the other hand, gun No. 2 was disabled by a crack which began at the right front of the slot and disappeared a little in front of the trunnion. The crack at its origin (near the slot) was about 0.016 inch wide, and penetrated the entire wall of the mantle. The gun-body, however, was uninjured; it had withstood the enormous enlargement of 0.171 inch (horizontal) and 0.138 inch (vertical) without any sign of cracking.

So far as our knowledge extends, this has never happened before in the extreme proof of a steel gun. Such results are met with only in bronze guns, or possibly in wrought-iron guns. As evidence of the excellent qualities of Bofors steel as a gun-metal, this 3.3-inch piece has been placed in the Ordnance Museum at Stockholm. As already stated, the 3.3-inch gun No. 3 showed no injuries; it may be used without danger for extended firing trials. As to the 4.6-inch gun No. 2, there is little to be said. It shows considerable enlargement of the chamber—much larger than that observed in the 4.6-inch gun No. 1, of softer steel and weaker construction. This is explained by the fact that gun No. 2 was fired most rounds with the very quickest powder delivered during the years 1880, 1881 and 1882, while for gun No. 1 the slowest powder of 1887 was used. No enlargements were observed at the rear of the chamber, thanks to the favorable action of the reinforce-band.

Besides these tests, the Swedish navy has tried a 4.7-inch ship-gun, selected at haphazard from a delivery. This was a banded piece, the tube alone being of Bofors steel. The hoops were puddled steel, the gun weighs 4167 pounds, the charge 16 pounds, the standard projectile 48.4 pounds. Up to the present time the piece has been fired eighty-six rounds, about one-half with 17-pound charges, giving an initial velocity of 1723 feet, an energy, total 998 foot-tons, and per pound of gun's weight, 1179 foot-pounds, with a pressure of 41,100

pounds. During the practice no scoring has occurred and no enlargement of the chamber is perceptible.

Most of the figures given above are drawn from the official reports.

From what has been stated, it is seen that the Swedish authorities have submitted the seven Bofors steel guns to exhaustive experiment, which has demonstrated that this gun-metal is perfectly reliable. This extremely favorable conclusion is due to the pains taken both by the army and the navy in carrying out the tests. The Swedish Ordnance Department, especially, has done everything in its power to facilitate these trials, so important to the country.

The Bofors plant is now in condition to execute large orders for guns, howitzers and mortars (at present, maximum length 17½ feet, maximum weight 35,000 pounds), as well as for gun-carriages, limbers, implements, ammunition-wagons and appropriate supplies. In sending orders, they should be accompanied, either by the necessary drawings, or by proper specifications, in which latter case the Works will make the drawings subject to approval. Thanks to its economical method of manufacture, the establishment will do work at very moderate cost.

NOTE.—Since the above paper was presented, the *Aktiebolaget Bofors-Gullspång* write me under date of March 7, 1888, in reply to my inquiry as to the feasibility of applying their process in fabricating heavy guns: "As regards the size of gun to be made according to our system of solid, non-hammered steel, there is no other limit than what is given by the scope of the machinery and the lifting power of the cranes!"

THE BEDDED ORE-DEPOSITS OF RED MOUNTAIN MINING DISTRICT, OURAY COUNTY, COLORADO.

BY G. E. KEDZIE, M.S., OURAY, COLORADO.

(Boston Meeting, February, 1888.)

THE ore-deposits of all that portion of the San Juan country within the borders of Ouray County are either in the tertiary eruptives or, confined to a relatively narrow zone, in the sedimentary beds just beneath them. The ore-deposits in the igneous formation

would be classed as fissure-veins. In addition to these, however, there is that form of fissure locally called "chimneys," of which the deposits in the National Bell, Yankee Girl and Silver Bell mines are good examples. In the sedimentary formation the ore-deposits are either fissures, usually along fault-planes, or intercalated beds between limestone, which forms the foot-wall, and a quartzite or shale that forms the hanging-wall. Of the bedded or contact-deposits there are two examples in Ouray county. One, known as the Mineral Farm, is just south of the city of Ouray; the other, known as the Red Mountain deposit, nearly six miles south from Ouray, will be considered in this paper.

Fig. 1 is a map, and Figs. 2, 3, 4 and 5 are sections, illustrating the facts stated below.

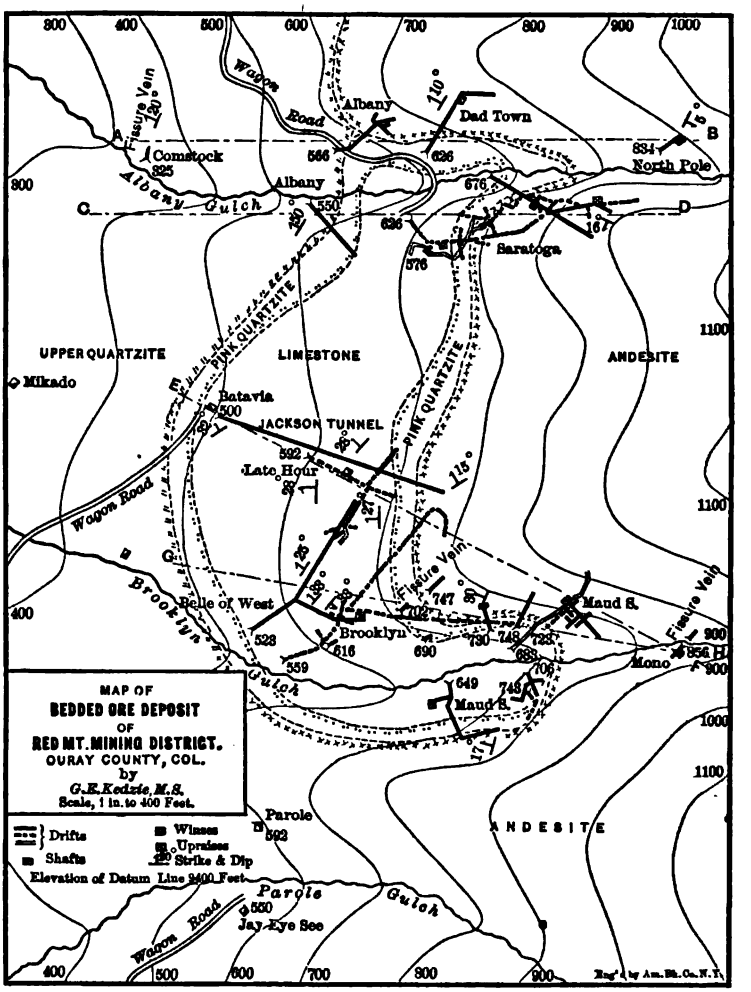
The tertiary eruptives, which are almost exclusively andesites, have a present maximum thickness of nearly 6000 feet; but previous to the Glacial epoch this formation, then forming a great plateau, must have been much thicker. The base of the andesites, resting unconformably upon the sedimentary beds beneath, has a mean elevation of 9000 feet above sea-level and an apparent slight dip toward the northeast. U-shaped valleys and cañons have been eroded in, and even through, this igneous rock into the sedimentary beds beneath, where the rounded and polished quartzites, with parallel *striae*, still furnish record of the genesis of the present orographic features of this country.

Leaving Ouray, which is at the altitude of 7700 feet, in a due south direction, we pass at the corporate limits a massive light gray, non-magnesian limestone, from 350 to 400 feet thick, dipping northwest at a low angle, and resting unconformably upon the quartzites beneath. This limestone, which I will provisionally designate as Lower Carboniferous, contains, in the upper portion of the bed, at its contact with the overlying shales, the Mineral Farm ore-deposit. This bedded deposit, which is worthy of more extended and systematic development than has yet been bestowed upon it, consists essentially of a barite gangue, with galena, carrying silver, undoubtedly in the state of sulphide, and occasional bunches of argentiferous tetrahedrite.

Passing on south over the wagon-road which has been blasted into the steep and nearly naked walls of the Uncompahgre cañon, we find the formation beneath the limestone to have a strike of N. 75° E., becoming east farther south, and an average dip of 60° northwesterly. This formation, at its upper exposure, consists of

dark-colored quartzite conglomerates, with a prominent band of laminated shale. These beds are succeeded by massive light-colored vitreous quartzites, which at the base become darker-colored and have interstratified beds of laminated shales. The exposed base of

FIG. 1.



this formation is at the center of a steep anticlinal fold crossing Red Mountain creek nearly 3000 feet above its junction with the Uncompahgre. This fold has an east and west trend and becomes broader towards the east. The vertical thickness of this quartzite

formation, as here exposed, is not less than 13,000 feet. It unquestionably belongs to the Archæan age. Passing south from the anti-

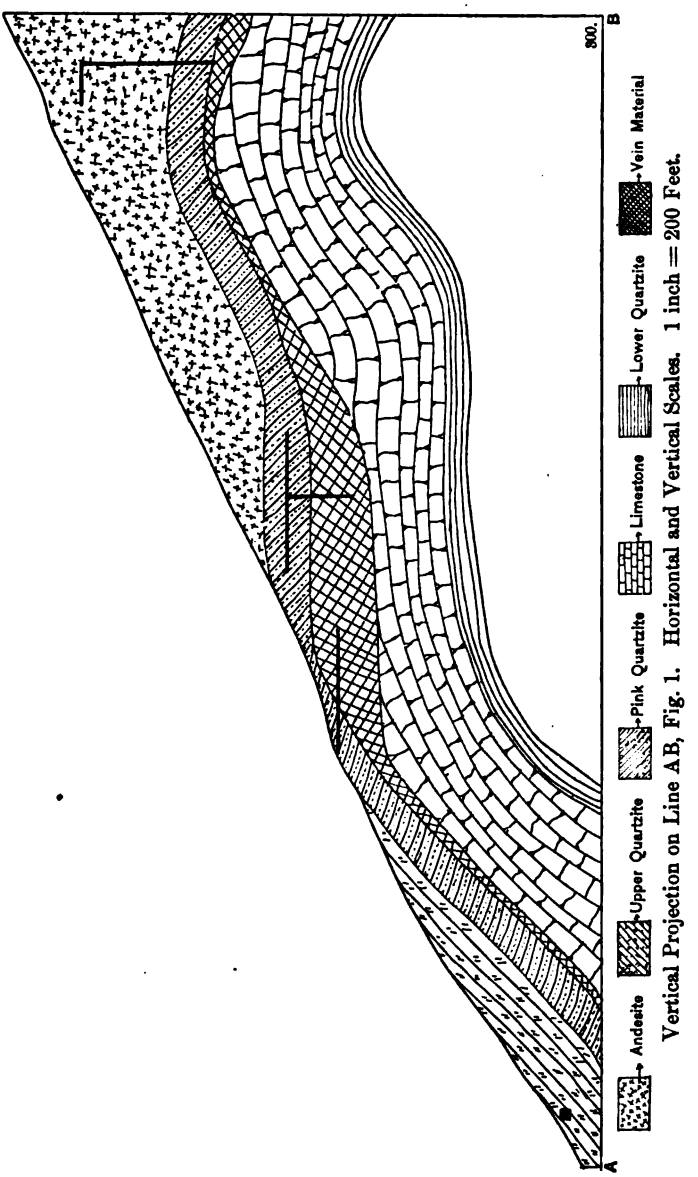


FIG. 2.

Vertical Projection on Line AB, Fig. 1. Horizontal and Vertical Scales. 1 inch = 200 Feet.

clinal fold, the strike of the rock soon becomes S. 45° E., with a dip of 60° southwest, and at a distance of about 3000 feet the formation

altitude of 9100 feet. About
ude of 9500 feet, there is the

Vertical Projection on Line CD, Fig. 1. Vertical and Horizontal Scales, and conventional Signs, the same as in Fig. 2.

e of the outcrop of an isolated
ain the Red Mountain bedded

deposit. This ore-body is on the west slope of Mt. Abram, near its base, between Brooklyn and Albany creeks, and at the center of the east side of the widened valley locally known as the Park. This park, which is nearly level, has several small shallow lakes containing quite extensive deposits of bog-iron. This iron, which is daily accumulating, is undoubtedly formed by the oxidation of the sulphate of iron held in solution by the water that has passed through the beds of decomposing pyrites which, as there is every reason to believe, underlie the Park at no great depth.

The outcrop of the ore-deposit is exposed by erosion in Brooklyn and Albany gulches, and along the west slope of the mountain, at an elevation of about 500 feet above the park. The limited exposure of the limestone indicates an anticlinal fold, much sharper upon the west slope, the axis of which has a northeasterly trend. Upon the summit and east side of this fold, the Upper Quartzite appears to have been entirely eroded, previous to the flow of the andesite which rests unconformably upon the sedimentary beds.

Upper Quartzite.—In descending series, from the park-level, the first formation encountered in passing up-hill is a white and compact vitreous quartzite, in beds from 1 to 3 feet thick, with a northwesterly dip of from 15° at the Park to 50° at the base of the bed near the south workings of the Albany.

Pink Quartzite.—This is a light-red argillaceous quartzite, in thin beds, having a maximum thickness of 35 feet. The top of this bed is a fine-grained conglomerate, consisting of worn fragments of white quartz, with a red argillaceous ground-mass. The base of the bed, which forms the hanging-wall of the ore-deposit, is finer-grained and lighter-colored, and, in the vicinity of the Maud S. workings, on the north side of Brooklyn Gulch, has been changed into a cherty conglomerate, while along the north side of Albany Gulch, in the vicinity of the Albany and Dad Town workings, the coloring has been almost completely removed and, in the cavities formed along the bedding and fracture-planes, there has been a re-deposition of crystalline silica and an occasional isolated bunch of fine-grained argentiferous galena.

Crystalline Limestone.—This is a white, massive-bedded, crystalline limestone, having a mean thickness of 140 feet. It is coarsely crystalline at the top of the bed, has a light gray band near the center, and is thinner-bedded and more siliceous at the base. This limestone forms the foot-wall of the ore-deposits, which are mainly located in the upper portions of the bed, at or near its contact with

the overlying pink quartzite. I made two analyses of this limestone, with the following result:

Top of Bed.		Base of Bed.	
Insolubles,	2.30	SiO_2	5.76
Al_2O_3,	trace	Al_2O_3 (trace of Fe)	4 91
$CaCO_3$,	97.30	$CaCO_3$	87.55
$MgCO_3$	trace	$MgCO_3$	1.87
	99.60		100 09

Lower Quartzite.—This is a light-gray foliated quartzite, in beds from 1 to 2 inches thick. This formation has been penetrated by the Jackson tunnel for a distance corresponding to a vertical depth of 140 feet, showing no material change in the character of the rock. An analysis proved the composition of this formation to be

SiO_2	90.88
Al_2O_3	7.57
	98.45

Andesite.—This eruptive constitutes the entire mass of the rugged mountains in this vicinity, and has a present thickness on either side of the valley of nearly 3000 feet. Microscopically, it consists of a light-gray or pink ground-mass, containing an abundance of white crystals of an altered feldspar, together with a few grayish-green crystals of either hornblende or augite, Occasionally fragments are found enclosed, giving the bed the appearance of a breccia. These enclosed fragments rarely exceed a few inches in diameter, and are of a dark green ground-mass enclosing white crystals of feldspar. Specimens of this formation from different horizons were submitted to Mr. Whitman Cross, of the U. S. Geological Survey, for examination. He provisionally pronounces the rocks andesites. My analysis of this rock gave the following result:

SiO_2,	58.85
Al_2O_3,	21.01
Fe_2O_3,	6.47 (4.53 Fe)
MnO,	.36
CaO,	7.08
MgO,	1.94
Na_2O,	4.73 (trace of K_2O)
	100.44

Though the sedimentary beds are much contorted, especially the lower quartzites, only one fault was found; and this appears to be a

reversed fault, with an upthrow of 45 feet and a strike of N. 45° E. This fault is found on the north side of Brooklyn Gulch, at the Maud S. workings. It is along this fault-line that a drift was run in the earlier developments, and a quantity of copper sulphide, rich in silver, was extracted. I found no evidence of the continuation of the fault into the andesite above.

Manner of Occurrence of Ore-Deposits.—There are several vertical ore-bodies partaking the nature of fissure-veins, the principal one being the Mono vein, with a strike of N. 45° E. and a dip of 68° S.E. This ore-body is from 1 to 2 feet in width, and is a continuous body of pyrite and chalcopyrite, to the bottom of the 70-foot shaft. In general this ore is not very rich in silver.

A vertical vein of barren white quartz with a few disseminated crystals of pyrite is found just east of the Brooklyn workings. This vein has a strike N. 45° E. and a dip of 73° N.W. The walls of the vein, where in the igneous formation, are strongly kaolinized and heavily impregnated with pyrite. I was unable to find a downward extension of this vein into the crystalline limestone, where it would have been cut by the workings beneath.

Lastly, at the Comstock workings, in the upper quartzite, there is a 1-foot vertical ore-body, having a strike N. 15° E. The ore consists principally of galena, with some chalcopyrite and blende, which, with large quartz crystals, forms a lining to a closely connected series of vugs.

The ore-body of greatest commercial value occurs in the crystalline limestone, at its contact with the overlying pink quartzite. The minimum thickness of the vein material is 3 feet; but it extends downward into the limestone for a very variable distance, being nearly 100 feet thick at the Saratoga workings, while, in the Brooklyn workings, the vein-material reaches nearly to the base of the limestone. The greatest longitudinal extent of these bodies of vein-material is in a direction parallel with the strike of the enclosing rocks.

The rich ore-bodies in the vein-material are very irregular, and the line of demarcation between rich ore and low-grade vein-material is exceedingly indefinite. There is, however, an apparent tendency of the richer argentiferous iron-ores to assume a horizontal position, while the light-colored kaolin bands with which the lead-values are associated assume a position nearer vertical along the original fracture-planes of the limestone. The pyrite and the great

mass of vein-material is of low-grade in silver, containing from a mere trace to 20 ounces of silver per ton of 2000 pounds. The

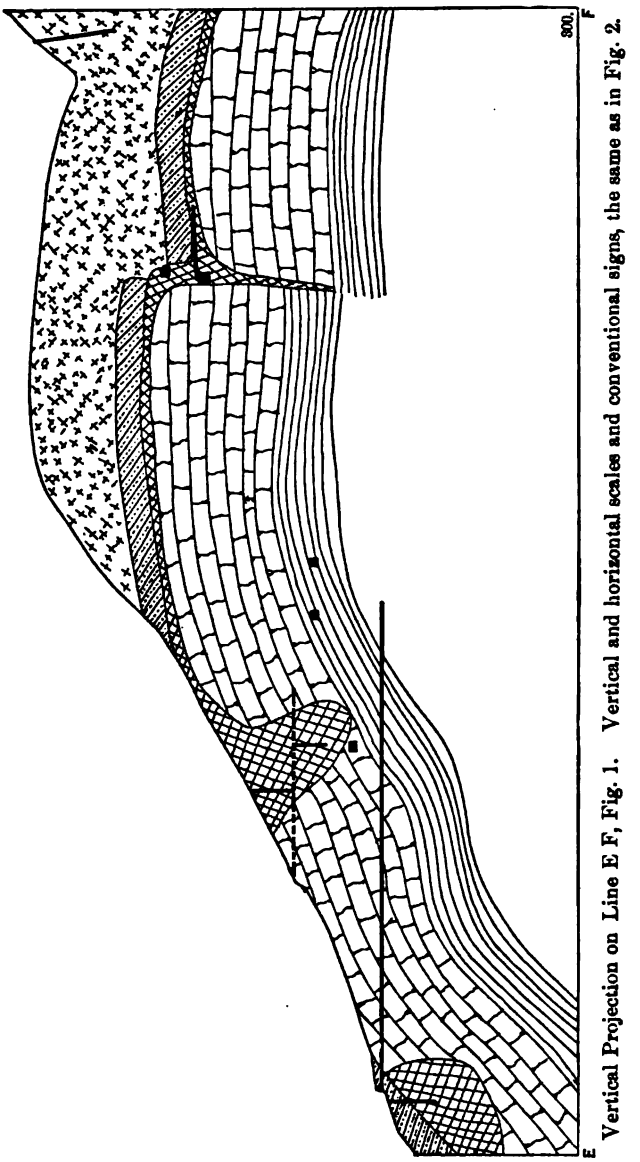


FIG. 4.

Vertical Projection on Line E F, Fig. 1. Vertical and horizontal scales and conventional signs, the same as in Fig. 2.

assorted ores, which are shipped to the smelters in 10-ton lots, vary from 50 to 80 ounces of silver per ton, with an occasional shipment

giving returns much higher in silver. All of the ores carry from 1- to 3-tenths of an ounce of gold per ton.

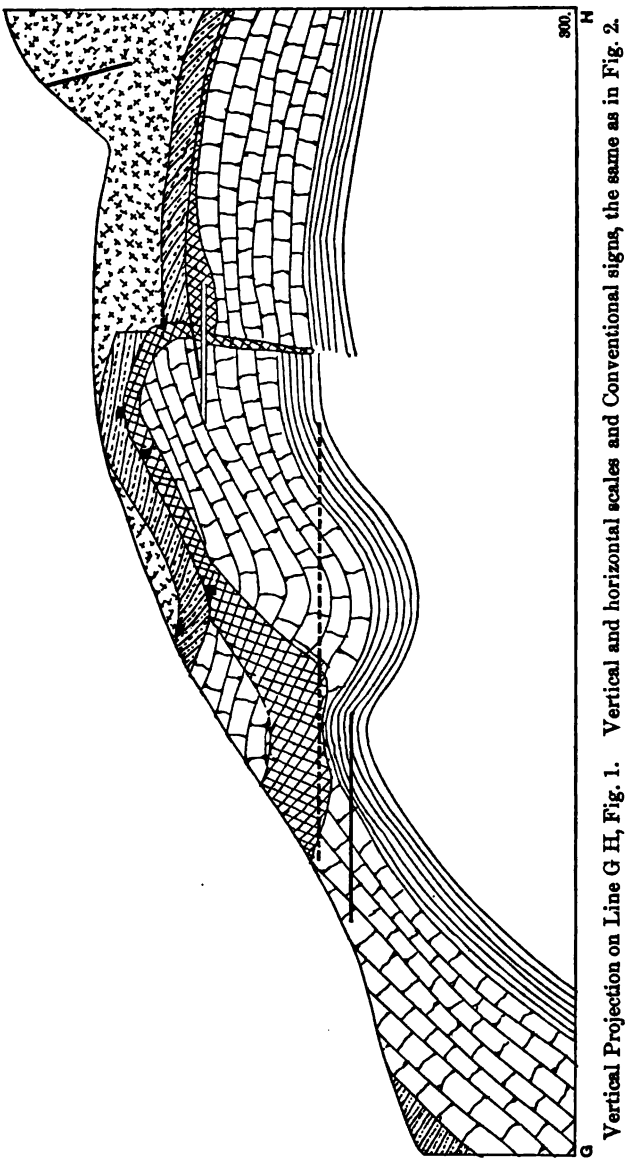


Fig. 5.

Vertical Projection on Line G H, Fig. 1. Vertical and horizontal scales and Conventional signs, the same as in Fig. 2.

Chemical Composition of Ores and Vein-Material.—Silver exists here as a sulphide; rarely in the native state. Occasional masses of

nearly pure sulphide are found in the oxidized ores. One specimen gave me returns of 21,430.6 ounces of silver per ton. Careful analysis failed to detect any appreciable amount of silver in combination with chlorine.

Gold exists undoubtedly in the native state.

Lead is occasionally found as a fine-grained galena, usually in the pink quartzite at the base of its bed. It may possibly exist to a limited extent as a carbonate or oxide, but the larger portion is found as a sulphate.

Copper exists as chalcopyrite, associated with the iron pyrite. Occasional lenticular masses of the black sulphide are found in the oxidized ores. Native copper is rarely found. Blue and green carbonates of copper are occasionally found as an incrustation.

Iron, the essential carrrier of the silver-values, exists at the base of the larger masses of vein-material as a sulphide. At the point where present oxidization begins the pyrite becomes granular and contains an abundance of iron-sulphate. The bulk of the iron, however, exists as an earthy, red and yellow, hydrous sesquioxide, though occasional masses of black anhydrous sesquioxide are found nearer the surface.

Manganese occurs as a dark-brown hydrous oxide, occasionally carrying up to 100 ounces of silver per ton of ore.

Zinc is found in small amounts, as a sulphide in association with the galena; also, as a light-brown hydrous silicate.

Silica is distributed throughout the vein-material in irregular masses and bands of friable granular quartz.

Alumina occurs in chemical combination with silica as hydrous silicates, usually contaminated by the iron and manganese oxide. A selected specimen of one of the typical varieties was analyzed, with the following results:

Al_2O_3,	39.07
SiO_2,	23.85
H_2O,	34.93
	97.85

Calcium sulphate is occasionally found disseminated through the vein-material as a secondary product, in shape of slender, transparent crystals.

Barium sulphate has not been found in any of the workings.

Ammonium oxide is usually present in the decomposed vein-material; in one instance reaching three-tenths of one per cent.

I made analyses upon two characteristic car-load samples of ore. No. 1 was of a light-gray color, carrying 128 ounces of silver and 0.22 ounce of gold per ton of ore.

No. 2 was of a light-red color, carrying 68 ounces of silver and 0.1 ounce of gold per ton of ore.

	No. 1.	No. 2.
H_2O (by ignition),	6.75	11.94
Insoluble silicates,	41.63	8.65
PbO,	18.40	...
Fe_2O_3,	17.12	64.80
Al_2O_3,	6.08	13.05
MnO,	.59	1.98
CaO,	1.70	...
ZnO,	1.23	...
SO_3,	3.87	...
CO_2,	2.39	...
	99.76	100.42

Product.—Very few of the mines, excepting the Saratoga, have reached a state of development that will warrant their being termed anything but "prospects." In the two years since the discovery and more intelligent development of these mines, there have been shipped about nine hundred and thirty-five tons of ore, having an average value of seventy dollars per ton. With the ground now opened, the product during the year 1888 should reach, if not exceed, two thousand tons.

From the isolated exposure, the limited development and the absence of paleontological evidence, I am unable to arrive at a satisfactory conclusion as to the age of the formation enclosing this ore-deposit. I have no theory to advance as to its genesis, believing that the collation of many observations of similar deposits is necessary before justifiable conclusions can be drawn.

WESTERN KENTUCKY COALS AND COKES.

BY JOSEPH H. ALLEN, MANNINGTON, KY.

(Boston Meeting, February, 1888.)

In the old Kentucky reports, made by the Survey of which Prof. Owen was director, the veins of the Western Kentucky coal-field were numbered from 1 up to 12. Later Prof. Shaler used letters, beginning at the top. For instance, Owen's 12 is Shaler's A.

Both systems are still in use; and hence, in order to be explicit, we are compelled to use both together.

Hopkins, Webster, Union, Henderson, Muhlenberg, McLean, Daviess, Hancock and Ohio counties are embraced wholly in the field, while Christian, Edmonson, Grayson, Breckinridge and Butler are partially underlaid. The following section is a generalization of about 40 local sections, besides the more general observations taken by Mr. C. J. Norwood, the State Inspector of Mines.

General Section of the Western Kentucky Coal-Field.

	Feet.	Inches.
1. Sandstone—"the anvil-rock,"		
2. Shale,		
3. Coal, A,	5	
4. Shale,		6
5. Limestone,	1	6
6. Shale,	0 to 2	
7. Coal, B,	6	
8. Under-clay,	2	
9. Sandstone	This space sometimes filled with a single massive sandstone.	25	
10. Coal (C?), not always pure,			3
11. Sandy shale and sandstone,		30	
12. Bluish shale,		25	
13. Bituminous slate,	2 to 4	
14. Pyritous band, filled with fossils (not always present),		2
15. Coal, D,	5′ 7″ to 4	8
16. Under-clay,	3	
17. Sandstone and shale,	45′ to 50	
18. Bituminous slate, containing fish-scales,	3′ to 5	
19. Coal, E,	16″ to 1	6
20. Interval, containing notably shales,	17	
21. Gray limestone,	2	
22. Interval, containing sandstone and shale,	10	
23. Black slate filled with fish-remains,	3	
24. Coal, E[a],	14″ to 1	8
25. Under-clay,	3	
26. Soft sandstone,	27	
27. Interval, sometimes containing red clay,	5	
28. Bituminous slate,	1	6
29. Coal, F,	9″ to 1	6
30. Under-clay,	3	
31. Mottled dark drab and ashy gray impure limestone,	1′ to 2	2
32. Shale,	20	
33. Sandstone and shale,	This space is sometimes filled with a massive sandstone 65 feet thick.	20′ to 35	
34. Coal, G,		1	2
35. Sandstone and shale,		30	
36. Earthy limestone,	6	
37. Dark calcareous shale; merges into No. 36,	7	6

	Feet.	Inches.
38. Impure coarse-grained limestone,		
39. Argillo-calcareous material,	1	
40. Black shale interlaminated with thin streaks of coal,		1½
41. Coal,	11″ to 1	6
42. Under-clay,	1	
43. Interval, mostly filled with argillo-arenaceous shale,	7	
44. Hard "cher" (chert),		6
45. Shaly sandstone,	10′ to 20	
46. Coal,	16″ to 2	7½
47. Sandstone, } 48. Argillaceous shale, }	varies from 70′ to 40	
49. Coal,	6	
50. Interval, 5 } 51. Sandstone, 35 } 52. Shale, 5 }	45	
53. Coal,	2	
54. Sandstone,	65	

Very little work has been done by the State in this western field, and hence we cannot speak positively of these veins. In many places they thin out, and in others some one or more disappear entirely. It will be noticed that I have not attempted to identify in the above generalized section, any beds lower than No. 34 (G). No. 53 is probably L, since, though in this section it shows only 2 feet thickness, L is certainly, at the point where we are engaged in working it, the lowest coal in the series. Commencing at the top, we have:

Coal A, or 12.—Area extensive but not greatly worked, the bed being near the surface and not in good working condition. In thickness it varies from 3 to 6 feet, and in character from firm, glossy, black and dense coal to rather soft and fragile. It cannot be used in furnaces raw, but cokes well.

Coal B, or 11.—Not trustworthy and full of slips. More persistent than A but has a clay parting. The top layer is a good gas-coal.

Coal D, or 9.—Usually hard and compact, with layers of fibrous coal; carries large percentage of sulphur. Makes good, strong coke. Vein 4 to 6 feet thick, and extensively worked.

Coal E, or 8.—The quality is good, but the bed is sometimes entirely absent. Where found it varies from 1.3 to 1.8½ feet in thickness.

Coals F, G, H, I and J are not of great value.

Coal L.—Is extensively found, though worked only in few places, and bids fair to be the best coal in the series. It is the vein worked

at Mannington, where the coal coked is giving a coke of excellent physical structure but somewhat high in sulphur. It is extensively developed in the counties of Edmonson, Grayson and Butler, where it runs from 3 to 7 feet in thickness, carries but little sulphur, and cokes well.

The above are the principal coals and their general characteristices.

Concerning their relations with the Pennsylvania series, Mr. Norwood, the State Inspector, says: "The present Geological Survey does not believe any seam in western Kentucky to be the equivalent of the Pittsburgh bed. The old Survey, or one or two of its officers, believed for a time that No. 9 (D) occupied the same geological horizon. Afterwards, Lesquereaux endeavored to show that No. 8 (E) was the one; while Dr. Owen thought it must be a seam higher than No. 9." The truth is that it is not possible to make out any equivalency between the sections of western Kentucky and Pennsylvania.

It is claimed by some of the State geologists that the Mud River Coal is the L vein; but from a personal knowledge of the L vein as worked by me at Mannington and also of the same coal elsewhere in the field and of the Mud River vein, I assign the latter coal to J, or No. 3. It is by far the most beautiful and purest coal mined in western Kentucky. The area of this bed is limited. It is about 42 inches thick, and the middle 36 inches is a beautiful black, glossy coal, very hard but brittle, and can be rubbed over a piece of white silk without leaving the slightest trace.

Running in an irregular east and west direction across this coal-field is a disturbance, which has divided the basin into two parts. North of this disturbance the upper coals are in place, while south of it the upper coals have disappeared, and the lower veins from 1 to 5 only are found. In the northern part of the field the following mines are worked:

Hecla,	Hopkins	County.
St. Bernard,	"	"
Central Coal and Iron Co.,	Muhlenberg	County.
Dovey,	"	"
Hamilton,	Ohio	"
McHenry,	"	"
Co-operative,	Hopkins	"
Taylor,	Ohio	"
Memphis Coal Co.,	Muhlenberg	"
Crabtree and Shotwell Mines,	Hopkins	"

While south of the disturbance are worked the

Clifton,	Hopkins County.
Empire,	Christian "
Mud River,	Muhlenberg County.
Mining City,	Butler "

The most important coals in the northern part of the field are Nos. 12, 11 and 9. In the upper veins the sulphur exists in an organic state, while in the lower veins 1, 3, and 5, it is in the form of pyrites. While there are some 12 or 15 mines worked, yet with the exception of the St. Bernard Coal Co., at Earlington, the output is small and both the coals and the iron-ores of western Kentucky are undeveloped. And yet there is no section of the South which offers any greater inducements to capital than this. Green river, which is navigable all the year round for boats drawing from 3 to 5 feet of water, affords an outlet to the Ohio and thence to the Mississippi. That section not adjacent to Green river is penetrated by the Nashville and St. Louis division, the Memphis division and the Owensboro and Nashville division of the Louisville and Nashville R. R., and also by the Chesapeake and Ohio—a part of the Huntingdon system. The section is nearer Chicago and St. Louis than the Connellsville or New river districts; and on coke alone there is a difference in freight-rates in its favor of from $1.50 to $2.50 per ton.

Various attempts have been made from time to time to coke these coals, with the following results:

The first attempt to coke any of the coals of western Kentucky was made in 1857 at the Airdrie furnace, situated on Green river in Muhlenberg county. This furnace was built by the Alexanders of Scotland, who having found what they believed to be the equivalent of the Scotch blackband iron-ores, and an associated coal, which it was thought could be used raw for smelting, erected a furnace after the Scotch pattern. The building of this furnace, which was run only six weeks and then blown out, and has since remained idle, has done more to retard the growth of western Kentucky than anything else. A full history of this enterprise is found in volume D. of the Kentucky Geological Survey. I condense from this account and the comments of Mr. Calwell, Geologist of the Survey, the following summary, in the hope of dispelling the error into which the failure of this enterprise has permitted the public to fall, and because the *Transactions* of the Institute will reach many more readers than can find access to the Kentucky reports.

The Airdrie furnace was finished in 1856. Its interior lines were: height, 50 feet; bosh-diameter, 17 feet; throat-diameter, 11 feet; height to bosh, 24 feet, shape cylindrical for 6 feet above bosh; hearth elliptical, 7 feet 4 inches by 5 feet, and 4 feet high. The furnace was open-topped, did not save the tunnel-head gases, and required separate firing for boilers and hot-blast. The two hot-blast ovens were of the old-fashioned pistol-pipe pattern. The whole plant was thoroughly built and still stands, with blowing-engine, etc., in good condition.

The large property (17,000 acres) contains 4 workable coal-seams: No. 12 (2 feet clear coal, 2 feet "brashy" coal, and 4 to 14 inches of blackband iron-ore above the coal); No. 11 (21 feet below No. 12, 6 feet of coal in 3 benches, of about 2 feet each, with 1 to 2 feet of pyritiferous shale between); No. 9 (84 feet under No. 11, and 19 feet below the level of Green river, 5 feet thick); and No. 5 (341 feet below No. 9, 3 feet 6 inches thick).

As to the coal, the following analysis will show its quality:

	I.	II.	III.	IV.
Moisture,	7.06	4.70	3.60	3.60
Volatile matter,	30.84	30.60	31.40	38.70
Carbon,	58.70	58.80	58.50	53.70
Ash,	3.40	5.90	6.50	4.00
Sulphur,	0.789	1.455	1.438	3.158
Specific gravity,	1.59	1.332	1.278	1.274

I., II., and III., No. 12 coal; IV., No. 11.

The following are analyses of the iron-ores made by Dr. Peter and Mr. Talbott:

	I.	II.
Peroxide of iron,	63.048	59.810
Alumina,	5.290	2.972
Brown oxide of manganese,	.090	.720
Carbonate of lime,	.680	2.263
Magnesia,	.930	4.270
Phosphoric acid,	.147	.223
Sulphur,	.044	.065
Combined water,	12.430	.266
Silica,	17.250	29.880
Lime,	——	——
Specific gravity,	3.246	3.652
Metallic iron,	44.133	41.867
Phosphorus,	.064	.097

I., Ore from the bottom of the shaft; II., roasted ore from the stock-pile, 17 years weathering since roasting.

These analyses warrant the conclusion that the No. 11 coal contains too much sulphur for iron-making; that No. 12 is too "fat" to be used raw; and that the iron-ore is rich and pure enough to be safely used, but could be advantageously mixed with other ores.

These difficulties were enhanced by the almost exclusive employment of Scotch miners and furnace-men, without the slightest experience of American ores and fuels. After three years of construction, the furnace was started at last with No. 12 coal, ran a few days very badly, was blown out on account of an accident to the boiler, which was repaired and started again, and after 22 days more, during which neither the quantity nor the quality of the product was satisfactory, had to be shoveled out, by reason of an accident to the engine. Another start was made, with no better results. This time the cast-iron shaft of the fly-wheel broke, and the furnace was blown out "for good"—the third stoppage within six weeks. As it was believed that the experiment proved the coal to be unfit for use raw, a large amount of coke was made in expectation of another trial. But this trial never took place. Operations were suspended; and except the mining of No. 11 coal to a considerable extent for the Southern market, nothing has been done for thirty years. There is no reason to doubt that with coke as fuel, and other ores (which are abundant within easy reach) to mix with the blackband, iron might have been successfully made.

An analysis of the above-mentioned coke, made after 17 years' weathering, gave the following results:

Moisture expelled at 212°,	7.50
Moisture expelled at red heat,	4.20
Fixed carbon,	82.90
Ash,	5.40
	100.00
Sulphur,	0.642

The same fuel and the same ores are still here. And with the advantages which Airdrie possesses, I can see no reason why this property, which was abandoned 30 years ago through ignorance, should not be overhauled and remodelled, and yet made profitably productive. Even should the coke be too high in sulphur, the Green river country offers magnificent advantages for the making of charcoal iron. I traversed last summer a tract of 40,000 acres, heavily timbered and underlaid with two beds of iron-ore and a seam of coal 4 feet thick, with a water transportation on both sides

and the center of the tract within 15 miles of a railroad, and with a ridge running entirely through the whole property, forming a natural road-bed.

The next attempt to make coke in this region was at the St. Bernard mines in 1882. Mr. Atkinson, the manager of the St. Bernard Coal Co., built two beehive ovens and obtained with crude washing the following results:

From No. 9 Slack-Coal.

Carbon,	84.32
Ash,	13.71
Sulphur,	2.54

From No. 11 Slack-Coal.

Carbon,	81.29
Ash,	15.68
Sulphur,	2.54

The results not being satisfactory, on account of the high percentage of sulphur still in the coke, no more ovens were built, and the two then used were allowed to fall into decay.

About the same time, Mr. Dovey, at Mercer Station, on the Chesapeake & Ohio R.R., built 7 ovens and attempted to make coke from No. 11 slack, washed in a Bradley jig. The same trouble was experienced here with the sulphur, and these seven ovens have been idle ever since.

No attempt had been made to revive this industry or to experiment any farther with these coals, until I came into the field in 1886. In January of that year I was requested by the President of the Clifton Coal Co. to make an examination of their mine, located in Hopkins county, to advise them as to the value of the property, and to suggest what should be done to make it profitable. The coal had become known as the poorest domestic coal on the market, and the property had been worked for several years at a loss. Upon examination I found that the vein was No. 1 or L, and had been opened on its eastern outcrop, which readily accounted for the immense quantities of sulphur and slate found in the coal. The shaft at this point had been sunk in a bottom, where the covering did not exceed 30 feet. The vein averaged in thickness 5 feet 10 inches, with several slate and shale partings and a band of sulphur (iron pyrites) near the top. In the main entry running west from the shaft and under the hills, I found the slate partings to be

fewer in number and the band of sulphur to diminish in size. No systematic plan of mining had been followed, and both ventilation and drainage were imperfect.

From this examination I was satisfied that as the mine was opened westward the character of this coal as a domestic fuel would greatly improve, and (provided the sulphur and slate could be taken out) a fair coking coal could be obtained. With this end in view, we began a series of experiments. A correspondence was opened with Mr. Stutz of Pittsburgh, Pa., the patentee of the Stutz coal-washer, and a sample of the coal was forwarded to him for his examination and test. Mr. Stutz reported the result of his experiments in the following analyses of the unwashed and washed coals, and the resultant coke:

	Unwashed Coal.	Washed Coal.	Coke.
Volatile matter,	33.32	35 24	1.28
Fixed carbon,	47.45	52 53	83.14
Sulphur,	3.21	1.92	1.98
Ash,	16.02	10.31	13.60

and added in his report, "I believe that a good and strong furnace coke for smelting iron, capable of bearing any burden, can be manufactured from this coal, provided it is properly treated and prepared. The coal in its chemical parts comes near to the Pittsburgh coal and possesses life enough to get the heat up." An analysis by Dr. Safford, of Vanderbilt University, made about the same time from coke made at the mines, gave the following results:

Carbon,	83.42
Volatile matter,	1.62
Sulphur,	2.02
Ash,	12.94
	100.00

The ash analyzed as follows:

Iron,	2.74
Silica,	4.45
Alumina, lime and magnesia,	5.75

The Stutz washing-machinery was known at that time as the best in the market; and, before proceeding further, Mr. Stutz was invited to come down and make a personal examination of the coal. He made this examination in May, and again reported as follows: "The coal is very bituminous and is a good coking coal, as has been fully shown by the different samples and tests made at different

places. Although the amount of impurities in some places is large (sulphur in the shape of iron pyrites, and bony coal), it does not present any difficulties to separate these impurities from the good coal."

The outcome of all these experiments and reports was the decision by the owners of the property to open the mines farther westward, to put up a crushing- and washing-plant and to erect coke-ovens. I assumed the management of this work in May, 1886, and in June a contract was closed with Mr. Stutz for a complete set of his crushing- and washing-machinery, with a capacity of from 250 to 300 tons per day. It was to consist of the following parts:

One conveyor-table, one set of 4-rolls crusher, one elevator for unwashed coal, one set of 2-rolls crusher, screw-conveyor, classifying-drum, three washing-machines set together, one elevator for washed coal, and all the necessary shafting, pulleys and belts.

The new work was commenced on the 17th June and finished on the 19th December. Within that time a new shaft had been sunk to the coal, 49 feet deep, 8 feet by 18 feet in size; new entries had been driven and the following buildings had been erected: hoisting-engine-room, boiler-room, blacksmith-shop, carpenter-shop, pit-head and tipple, crushing- and washing-house and storage-bins; also 34 bee-hive ovens, 11 feet 6 inches in diameter.

The excavation for this battery of ovens, for the coke-yard and for the extension of the railroad switch involved the moving of 28,000 cubic yards of earth. A trestle-work, 310 feet long, 12 feet wide and 16 feet high, was also built to connect the coal-bins with the ovens. It was originally intended that the block of ovens should comprise 50; but the configuration of the ground and the lateness of the season compelled us to stop at the 34th, with the expectation of making the number to be built on the opposite side of the track 66 instead of 50, as the whole plant had been designed for 100. As fast as these ovens were finished, fire was put in them in order to season them, and by the 19th of December the whole battery was smoking.

The season was a wet one and all the material green; and it was not until March that the ovens were really hot enough to make coke that would stand transportation. Even then the coke was soft. Sale was found, however, for all the coke made; and as the conditions grew more favorable the coke improved, and the price advanced from $1.50 in March to $2.30 in May and June. In operating the property under this new condition of things, many difficulties were

encountered. The first great trouble was experienced in crushing the coal to the proper size; and the crusher had to be adjusted to meet this requirement. This difficulty has not been entirely overcome yet, nor will it be, until an additional pair of rolls is added to the crushing machinery. No coal should be permitted to go into the washer over three-eighths of an inch in diameter. If this were the case a much cleaner product would be secured from the washer and a harder coke from the ovens, lower also in ash and sulphur.

Much difficulty was experienced in the adjusting the washing-machinery to our coal. Mr. Stutz having left as soon as his machinery was set up, we had to find our way through this mass of machinery as best we could. The greatest trouble here was, to eliminate from the coal the very thin particles of slate which, owing to their light weight, floated on top of the clean coal, were scraped off with it and passed into the ovens. This slate appeared in the coke, causing it to break into very small pieces, and very largely increasing the percentage of ash. Mr. John Fulton, in reporting on this coke at that time, says: "The physical tests show a normally firm coke with rather large cellular structure, slashed with discs of slate which render it brittle and make its capacity for bearing furnace-burdens much under its real capacity. Its hardness of body is slightly below Connellsville; not so much, however, as to impeach its usefulness in this respect." An analysis of the coke at this time showed 17 per cent. of ash.

This was, however, at last overcome, and a change for the better became at once apparent in the coke, the percentage of ash having decreased to between 12 and 13 per cent.

When we first commenced to wash the coal, and even as late as June, we used the mine-water for this purpose; but upon analysis it was found that this water contained 127.03 grains of sulphates to the gallon. It was therefore abandoned, and an arrangement was made to receive and store surface-water for this purpose.

During the time this plant has been worked we have made about 5000 tons of coke, shipped principally to St. Louis, Chicago, Memphis, Louisville and Newport, Ky. It has been received with favor, and no complaints have been heard when the ovens were running continuously and a high temperature was maintained. The coking was usually done in 48 hours, except the charges which were put in on Friday and Saturday; these were drawn on Monday and Tuesday.

It has been impossible to tell what the exact loss in washing has

been, but from a careful estimate it cannot exceed 12 per cent. The yield in coke has been about 54 per cent. This compares very favorably with the yield in the neighboring coking-fields: Tennessee showing a yield of 51.1; Ohio, 56.4; and Illinois, 50.7 per cent.

As all former attempts in this section to make a satisfactory coke have failed, rather owing to imperfect washing than to the fault of the coals, much criticism was indulged in when our work was commenced. Encouraged, no doubt, by the success which has attended our efforts to produce a good metallurgical fuel, the St. Bernard Company has again revived its interest in the matter and is now engaged in building 25 new ovens. From time to time I have made various tests of nearly all the coals in the western field, and in all cases the coke has a bright silvery luster, a clear metallic ring and good physical structure. Sulphur seems to be the only drawback; and I am fully persuaded that this difficulty will be at last overcome. In a great many of the cokes, even now, the sulphur is much less than in some Alabama and Tennessee cokes which are extensively used by the furnaces of Chattanooga and Birmingham. I append a comparative statement of the analyses with the names of the chemists. With the development which is now going on in this field, western Kentucky will yet solve the problem of cheap metallurgical fuel for St. Louis and other western markets. Our rates of freight to Chicago, St. Louis and other western points are from $1.50 to $2.25 per ton less than those of Connellsville; and while it costs us more to produce a ton of coke than it does Connellsville or New River, yet the difference in freight more than compensates, and leaves a fair profit in addition. Although the analyses of the Alabama and Tennessee cokes show a percentage of sulphur equal to some of our western Kentucky cokes, yet iron-makers claim that in all our coals the sulphur is too high. Those of us who are interested reason from the developments that are now in progress that the day is not far distant when all the coals of this section can be compelled to give up this excess of sulphur when made into coke. The iron-ores of Lyon, Trigg, Caldwell, Edmonson and Muhlenberg counties are all within easy reach of this field, and the rail and water facilities, coupled with the solution of the coke problem will necessarily create a new Birmingham somewhere in this section.

Apart from the iron interest, Green river alone is fully able to supply all the lower Ohio and Mississippi river points with coal of as good quality as Pittsburgh, and at a saving of 700 miles in transportation. For a length of nearly 120 miles out of the total 225

miles from Bowling Green to Evansville, this grand water-way traverses the western coal-field; and thousands of acres of excellent coal are here yet undeveloped. Within the past two weeks I have been over a section on which I found No. 11 coal, 7 feet 2 inches thick, while 55 feet below is No. 9, 4 feet 6 inches thick.

One of the oldest and most experienced Alabama coal operators, when shown this property, could not conceal his delight, and is yet wondering why all the small veins of Alabama and Tennessee are developed and this has remained unopened so long.

No. of Sample.	Place at which Coke was obtained.	Name of Chemist.
1.	Mary Pratt stock-pile,	A. S. McCreath.
2.	Sloss Furnace ovens,	"
3.	Woodward Iron Co.,	"
4.	Williamson Furnace Co.,	"
5.	Citico Furnace Co.,	"
6.	" "	"
7.	" "	"
8.	Chattanooga Iron Co.,	"
9.	St. Bernard Ovens,	Mr. Morrell.
10.	"	"
11.	Made at Clifton Coal Co.'s ovens,	Dr. Peter, of Ky. Survey.
12.	" " "	" "
13.	" " "	" "
14.	Stock-pile at furnace,	Kentucky Geological Survey.
15.	Made at Clifton Coal Co.'s ovens,	Mr. Stutz.
16.	" " "	Prof. Safford.

	Name of Coke.	Water.	Vol. Matter.	Fixed Carbon.	Sulphur.	Ash.
1.	Pratt	1.92	7.58	88.875	1.182	8.993
2.	"	.130	1.130	86.478	1.049	11.213
3.	"	.180	.640	89.164	.670	9.346
4.	Coalburg Mine	.128	.685	84.678	1.879	12.630
5.	Etna	.856	1.160	85.45	1.451	11.08
6.	Daisy	.218	1.055	79.83	2.132	16.75
7.	Soddy	.172	1.098	80.82	2.127	15.78
8.	Dade	.542	1.091	75.94	.670	21.75
9.	St. Bernard, No. 9	.....	.615	87.275	2.21	12.110
10.	" No. 11	.....	.34	90.69	2.37	8.96
11.	Tar Lick, Ky	.40	.....	79.70	2.27	19.90
12.	Nells Fork, Ky	1.74	.06	81.60	1.56	16.60
13.	Sycamore, Ky	.80	.30	82.00	1.67	16.90
14.	Airdrie Furnace,					
15.	No. 12	7.50	.....	82.90	.64	5.40
16.	Clifton Coal Co.	.....	1.28	83.14	1.98	13.60
	"	.....	1.62	83.42	2.02	12.94

CERTAIN CONDITIONS IN THE MANUFACTURE OF STEEL RAILS, WHICH MAY GREATLY INFLUENCE THEIR LIFE IN SERVICE.

BY FREDERIC A. DELANO, CHICAGO, ILL.

(Boston Meeting, February, 1888.)

In adding one more to the list of papers that have been presented on the subject of steel rails, I wish to explain at the beginning that in enumerating the steps in the manufacture which, I think, have an all-important effect on the wearing-qualities of the rail produced, I do so with no little hesitation. As a student in the field, I am anxious for the opinions of others, and have finally decided to ask for it in this way. I have not stated everything which I think affects, in one way or another, the life of rails, but have confined myself to those steps in the manufacture which I felt most sure did do so. Recognizing the danger of being carried away by enthusiasm, I have endeavored to make my assertions with caution, and to understate my case rather than to overstate it; but if I have succeeded in throwing a ray of light on this vexed question and placed it in the eyes of some railroad men in an aspect which they have not fully considered, my boldness will be more than justified.

I. *Uniformity in the quality of the material charged into the converter assists the attainment of uniformity in the steel produced.*— This is the argument which is used by the champions of the cupola-melting or indirect process, and has considerable weight. It takes vastly more skill to make good steel if there is a great irregularity in the iron charged into the vessel than if there is not this irregularity; but on the other hand, there are certain advantages in the direct method, such as a lower sulphur-percentage from the same furnace-iron, while it is possible to get as great uniformity in the iron charged by having the furnace-capacity enough in excess of the demands from the converting department, to allow for occasional "slips" in the furnaces, and permit the mixing of the iron from two furnaces which are respectively too "hot" and too "cold."

II. *Care in pouring ingots at the proper temperature into warm moulds, and the treatment of ingots before rolling, have important effects on the rails made from them.* In other words, many of the failures in rails and the uneven wear of rails are often traceable to

bad ingots, which may result from careless pouring, spattering the sides of the moulds, or stopping partially or entirely the flow of metal from the ladle to the ingot. Cold moulds produce "flaws" and "checks" in ingots, and the use of moulds so hollowed and burnt as to require an hydraulic ram to push out the ingots also does it.

III. *Increasing the diameter of rail-ingots may be a positive injury.* —(*a.*) The pressure between the rolls should increase as the square of the diameter, if large ingots are to be forged as thoroughly as small; yet by driving the rolls more slowly their effect can be made to penetrate more deeply to the center of the ingot. (*b.*) The amount to be cut from the top of the ingot to reach soundness is by weight a certain percentage of the cross-sectional area, and therefore must increase as the square of the diameter. (*c.*) Increased height, within practical limits, gives an increased amount of metal that is sound, since the soundness of any part of the ingot should be proportional to the weight of the metal above it.

In these three ways, I think, the proportions of ingots have an important influence. First, that the power of the forging press, the rolls, or the steam-hammer must increase certainly with the area of cross-section or as the square of the diameter, and that, unless this is done, the forging effect is superficial and not thorough. With rolls this effect can be made to penetrate more deeply by simply allowing more time for the pressure to be transmitted through the semi-rigid mass, *i.e.*, by slower rolling. With high-speed rolling the effect is like that of a light, quick-striking hammer, and an index of its superficiality is to be seen in the hollow, cup-shaped ends.

Second comes the question of the amount necessary to cut from the top of the ingot to reach soundness, and how this is affected by changing the proportions of ingots. The distance from the top to which the central funnel-shaped cavity and dispersed honey-combs will extend (conditions being equal), is about the same, irrespective of the cross-sectional area, and hence is by weight proportional to this area, or varies as the square of the diameter.* It seems but

* At most mills, the bloom goes to the shears "top" end first, which seems to me so wasteful that I speak of it here. Even granting that the "bottom" end needs some trimming, it would be better to trim that first, and then cut off the amount left over at the top instead of cutting from the top end first, which not infrequently results in wasting more from the bottom end than has been cut from the top.

reasonable that if we keep the top of the newly-poured ingot fluid as long as possible (as by throwing on charcoal or coke-dust), and keep the ingot erect until the fluid interior has solidified (most easily done by heating in furnaces in which it stands in the same position as it was poured), we shall enable this cavity and general unsoundness to exist near the top; whereas, if by throwing on water or by "stoppering" we solidify the top of the ingot, we shall oblige the unsoundness to exist further down in the ingot. Third, it seems but a corollary to this proposition that the soundness of any part of the ingot is roughly proportional to the height of metal above it. This theory could not be pushed to extremes, since there are sound, practical reasons against making the mould too tall and slender.

IV. *Rolls intended for the blooming of ingots by rapid reduction at each pass** (1) *should be larger in diameter in proportion as the reduction per pass is larger; and* (2) *should be designed* (*as, for example, universal or "V"-shaped rolls*) *so as to support the metal on all sides during the rolling, and thus prevent the formation of little cracks and tears in the unsupported part.*

The first statement is reasonable and needs no explanation; and in regard to the second, I think I am speaking within reasonable bounds in saying that, as rolling is more rapid, or as the amount of reduction per pass is greater, the width of unsupported metal between the collars is more subjected to cracking and tearing. This trouble would be overcome, I think, by "V" rolls; but I confess I have never seen the experiment tried, and may be mistaken.

V. Rolling long-length rails certainly lessens the waste, and a mill rolling long-lengths can better afford to throw away a long rail-end at the top than a mill rolling short-length rails; but besides this, it influences the temperature at which rails are finished, though the coolness at which it is possible to finish rails of the existing type of section is really limited by their shape. This brings me to my fifth proposition—*that the hot rolling of rails necessitated by the present shape produces a rapidly-wearing rail.* This hot rolling and extremely hot finishing produces a metal loose in structure; and the head of the rail, particularly hot when finished and cooling very slowly, is on this account rapidly worn away as soon as the outer skin is worn off. Already the weight on the driving-wheels of

* The only way to compensate for the loss of time resulting from reducing the speed of rolls, even to one-fourth of the present speed, would be to decrease the number of passes; for instance, doing in 5 or 6 what is more commonly done in 10 or 12.

many locomotives is sufficient to "squash and flow" the metal in the rail-head of many of the rails manufactured now-a-days, and there are only two ways in which this trouble can be met—which brings me to my sixth proposition.

VI. *The only way to materially increase the life of rails involves a change in the ordinary type of rail-section, in order to* (1) *increase the bearing-surface between the wheel-tread and the rail, so diminishing the weight per square inch;* (2) *to increase the hardness or, more accurately, the compressive strength of the metal,* (a) *by increasing the compactness or physical hardness of the metal* (*only possible by colder rolling*), *or* (b) *by increasing the chemical hardness* (*only possible, with safety, in a section less liable to internal strains*).

It is evident that it will be of little use to increase the width of our rail-heads unless there is some change in the shape of locomotive tires; for, of the eighty or more different types in this country, only a few when newly turned up get a bearing across the entire width of present rail-heads, and the shape of many rails, taken out of track, attests the condition of the tires, and proves that the rails were worn out before their time while acting as grind-stones, on which the tires have had their "coning" worn down to an even bearing.

Many engineers have calculated the strength of their section on the assumption that the metal in all parts of the section was of equal strength, as if a rail had been planed out of a rectangular bar. What better instance that they have done so do we need than the experiments (so much talked of) made by the late Von Weber, of planing down a five-eighth inch web to ascertain what would be a safe minimum thickness? Much, too, has been sacrificed to making a rail for the joint, a sacrifice well worth making, perhaps, if a good joint be unattainable otherwise. The important thing is to know how much we are sacrificing, and how much we are getting for our sacrifice. I cannot but think that if engineers realized the sacrifice we should see a splice-bar devised which would be as good as the best and yet suited to the best form of rail.

In the question of hardness as affecting the life of rails, first brought into prominent notice by the careful investigation of Dr. Dudley, there seems to have been no distinction made between the two different kinds of hardness—that which results from greater compactness of the metal, which I shall call *Physical* hardness, and that which comes from the chemical composition, which I shall call *Chemical* hardness. Supposing the surface of a rail to be in reality

rough, resembling, when much magnified, a roughness like that on the soles of rubber over-shoes, it is conceivable that these roughnesses should be so brittle as to break off, but it is clear that the greater the physical hardness the greater will be the tenacity and resilience of these minute roughnesses. Physical hardness has, in fact, this distinction from chemical hardness, that it does not entail brittleness—is diametrically opposed to brittleness. Resilience and toughness are as necessary to increase the life of rails as they are necessary for safety, yet it follows, that for a given toughness, resilience, or elasticity, the greater the hardness the greater will be the wearing capacity. Hardness due to chemical composition is, we say, coupled with brittleness; but it will be admitted that, for any given chemical hardness, the brittleness increases very rapidly as we approach a shape liable to internal strains, and, the limit of brittleness being the "danger-line," chemical hardness is more possible as we approach a shape of section not liable to internal strains. As I have already said, the weight on engine-drivers is even now sufficient, with the bearing-surface attained, to "squash" and "flow" the metal in the rail-head of many of our rails. Increased strength to resist these compressive strains is given by greater hardness, physical or chemical; but may we not have reached a point where a greater chemical hardness would be unsafe, yet greater physical hardness impossible, until we have a section which will admit of slower rolling and colder finishing?

VII. *Rails, if anything but straight, should be a little low in the center and not cambered.* Many rails go into track with a perceptible camber in them, a practice which has grown up, either because the supports on which the rails are inspected are more than 15 feet apart, or because of a notion of rail inspectors that a rail, like a bridge, should be slightly arched. This, of course, is absurd, and if anything but straight it would seem better that the rail should "sag" slightly in the center to partly compensate for the lengthening by service of the upper fibers of the beam.

VIII. *Our present knowledge does not justify us in stating the composition of steel which will give the best-wearing rail for the least money, but it seems probable that sections permitting colder rolling and less liable to internal strains will safely allow greater latitude of composition, and yet have good wearing-qualities.*—So many men believe that if we could analyze every rail we should at once know how it would wear and all about it; that to them the consideration of the mechanical conditions of manufacture seems useless. The fact is, that analyses

of rail-steel tell very little, and a man could about as well judge the value of a piece of granite as a building-stone from knowing the exact amount of silicon in it, as he could judge the quality of a rail from knowing the amount of carbon, phosphorus, silicon, manganese and sulphur in it. Metallurgical chemistry has made and is making great strides; but, as chemists will admit, they cannot yet tell how these elements are combined with the iron, whether they influence the whole "matrix" or whether they are combined wholly among themselves as particles of foreign matter, local in their effect and insignificant. We know more about the effect of carbon than that of any other metalloid which combines with iron in steel, and we know that its condition is greatly affected by sudden or unequal changes of temperature. May it not be so with the other elements? Since we know that phosphorus produces a tendency to coarse crystallization, and that it is probably more potent than carbon in producing internal strains, may we not infer that, if it be important to avoid internal strains and slow cooling from a high heat with carbon-steel, it is doubly important to avoid it with steel containing notable amounts of phosphorus?

(The all-important consideration of getting a rail-section admitting colder rolling will not necessitate our going back to the old pear-shaped section, but it probably will require a slight thickening of the web, increasing the slope-angles above and below to say 18 degrees, and increasing the fillets connecting the web with the base and head to three-fourths of an inch, limiting the height and width of base, for a rail of 75 pounds or under, to 4½ inches.)

A vast amount of money has been thrown away in making analyses and physical tests of rails about which little or nothing was known, and after the study of records upon records of analyses which only seem intended to bewilder and confuse, I feel justified in saying that a small part of this money judiciously spent would have given us an immense amount of valuable information which we now lack. If every large railroad company in this country should begin now, and, choosing one rail in every thousand tons ordered, make some practical service-tests, we should begin in a couple of years to learn something about our subject. The clause which I would suggest including in rail-contracts would be: "The representative of the railroad company shall pick out one rail for every 1000 tons, or fraction thereof, ordered (on which is distinctly stamped the heat number), and this rail shall have one foot in length sawed off cold from each end. The 28-foot length shall be *daubed* (on the web

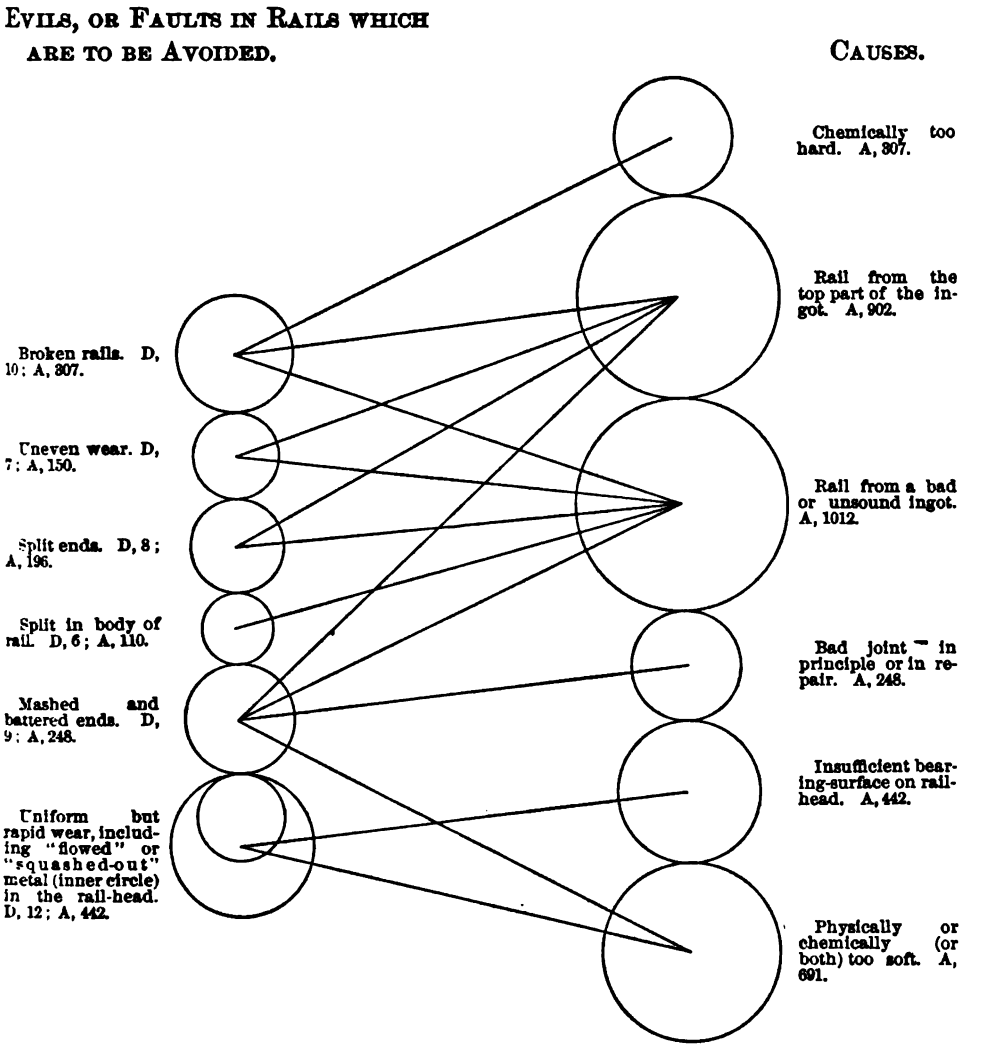
and flanges) conspicuously with paint, and the short pieces stamped so that it will be known without possibility of error from what rail they have been cut, and to which end they belong. The painted rails and the short pieces shall be kept together by the rolling-mill company until the completion of the order and then forwarded to the railroad company."

My idea would be to test all the 28-foot (painted) rails of one year's make by putting them at one time into a piece of track where they would have practically identical service, and where they could be watched. Any additional refinements which could be added—to record the numbers of trains, the tonnage, etc., would be of great use; but even without them much could be learned by simply watching the comparative wear.

The short pieces could be tested in various ways, either at the outset or after the track-test was complete. An etching of a planed section would show roughly the part of the ingot from which that part of the rail was rolled, and there would also be the opportunity for making tensile and torsion-tests with specimens prepared from the head, web and flanges, and chemical analyses from the specimens themselves at their point of fracture, which would give us some right to compare physical properties with chemical composition.

The following chart will help to illustrate some of the points I have indicated. It is intended to show in a graphic way, first, the evils or faults in steel rails which are to be avoided; secondly, the causes which are accountable for these evils.

The circles on the left indicate roughly by their magnitude the relative importance of the various evils named. To express this estimate, diameters of 10, 7, 8, 6, 9, 7, and 12 respectively have been taken (the unit on the actual scale being one-sixteenth of an inch), and the corresponding areas, given in thousandths of a square inch, indicate the estimated relative importance of the respective evils. (Of the justice of this estimate, others must form their own opinions.) The lines connecting the two columns of circles indicate the causes which may produce each evil; and the magnitude of the right-hand circles, representing the relative importance of the "causes," is so calculated that the area of each cause-circle is the sum of the areas of the evil-circles with which it is connected. The most important causes to be avoided in manufacture are thus seen to be those which may produce the greatest number of occasions of failure, and the importance of the causes is proportional, on the assumptions made, to the areas of the right-hand circles.



Graphic Representation of the Relative Magnitude of Evils or Occasions of Failure in Steel Rails, and the Consequent Importance of the Causes of these Evils.

D = Diameter in sixteenths of an inch.

A = Area in thousandths of one square inch.

AN IMPROVED SYSTEM OF WATER SUPPLY FOR HYDRAULIC MINING.

BY H. D. PEARSALL, LONDON.

(Boston Meeting, February, 1888.)

IT is well known that the usual system for supplying water at high pressure for the purposes of hydraulic mining possesses serious disadvantages in expense, delay and large annual repairs. Where plough work is admissible, the first expense of the ditch is moderate; but it is very often three years before the work is solid and the ditch water-tight, heavy expenditure for repairs being needed during this time and great loss of water occurring. Moreover, in the course of a long ditch, it usually happens that part of it has to traverse a difficult country where fluming, masonry, or other expensive construction is necessary, which makes even the first cost high. The life of flumes is also short.

The advisability of pumping water instead of supplying by gravity has therefore often been suggested, when sufficient water-power for the purpose of pumping happens to be available. But the expense of the requisite machinery and its complication, which involves skilled attention and expensive wear and tear, has hitherto prevented.

The author, however, having been called upon to make plans and estimates for hydraulic works for an extensive placer-mine in Colorado, has succeeded in devising pumping arrangements which are free from these objections, and are in fact estimated to cost less than half what a ditch on the usual plan would cost. Several incidental advantages in the plan also accrue, which will be referred to more particularly below.

The works on the placer-mine have not yet been carried out, on account of financial arrangements, but in the meanwhile an hydraulic engine, similar to that designed for the mine, has been constructed and set to work for other purposes in England; and in order to explain the plan which I have recommended for working the placer-mine, I will describe this, which is actually at work, in preference to describing the one which is not yet constructed.

The engine or machine acts on the principle of the hydraulic ram

to this extent: that both the new machine and the hydraulic ram "obtain their pumping power by the arrest of a column of water which has been previously set in motion by gravity."

I must, however, protest in advance that the new machine differs so much in construction from an hydraulic ram that the objections to which the latter is open are not applicable to the new engine. I am perfectly aware that the hydraulic ram cannot be made to deal with the quantities of water now in question; but, on the other hand, I have demonstrated by actual working that the new machine can do so.

The feature of hydraulic rams which has restricted them to a small size is their violence. In the new machine this is not only reduced but has no existence at all. It works with all the smoothness of a well-constructed reciprocating engine. This is best shown by indicator-diagrams taken from the pressure-chamber. Several of these are given in Fig. 2. These diagrams were taken with ordinary steam-engine indicators.

By using what may be called the fundamental principle of the hydraulic ram, we are able to do what we cannot do in any other way—pump water by the agency of water-power directly by one machine, instead of using turbines to obtain power, which turbines have then to drive pumps. Moreover this one machine is a far simpler one than the pumps alone which would otherwise be required, and as it works very slowly it requires much less repairs.

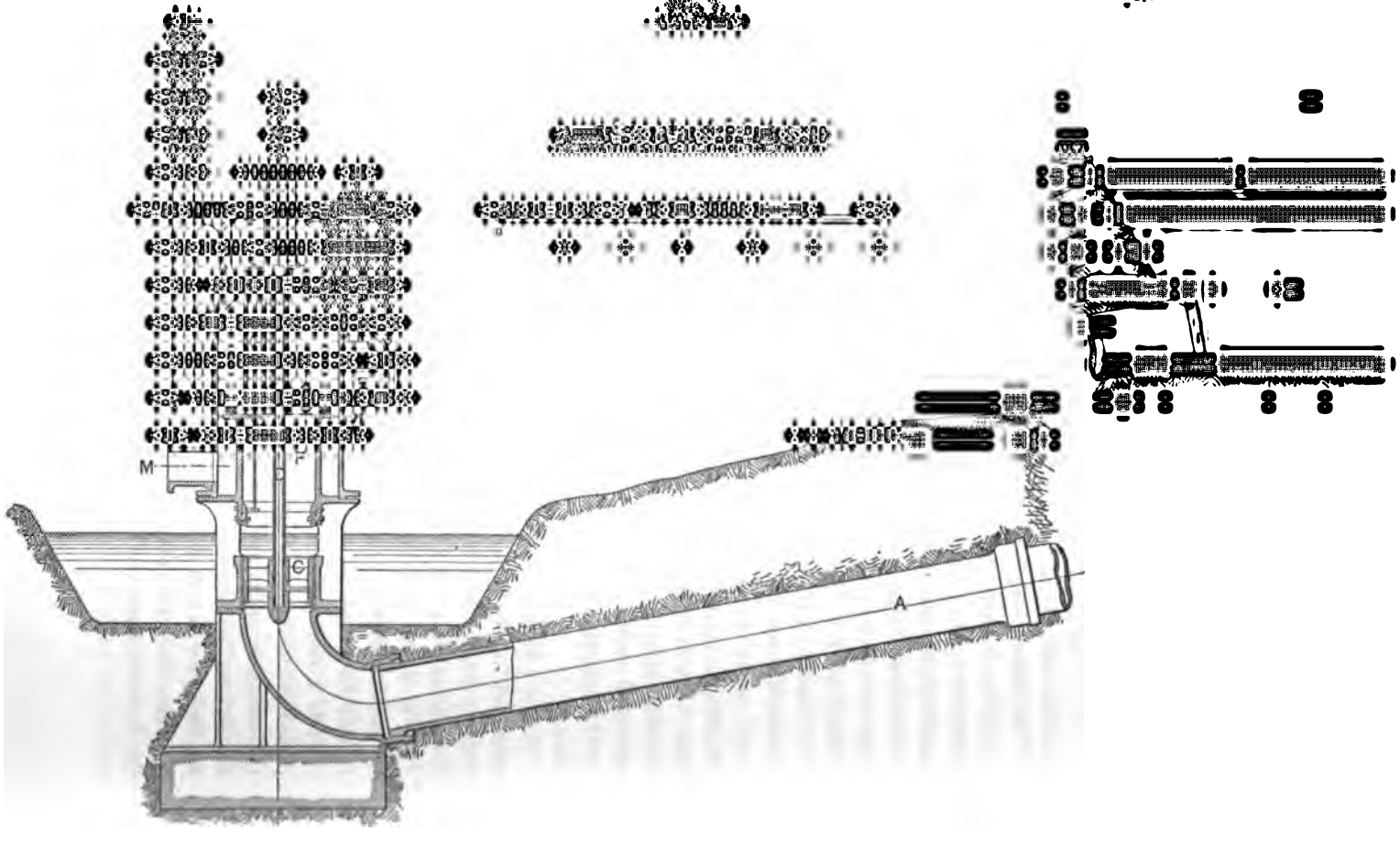
The construction of the existing engine is shown in Fig. 1. as follows:

A is a pipe (called a flow-pipe) which conducts water from the source of supply B to the engine. C is the main valve. The water flowing down the flow-pipe freely escapes through valve C, when it is open, as shown in the figure.

D is a rod attached to valve C, by which valve C is moved up and down at proper intervals of time by means of the motor E further described below.

F is a chamber or receiver immediately above valve C. At the period of the stroke of the engine which is represented in the figure, this chamber contains only air, and communicates freely with the atmosphere by the pipe G. At the base of pipe G, there is a valve J which carries a float.

When the main valve C is raised and closed, it of course shuts off the flow of water; but it does not interfere with the flow of the water


M
C
A

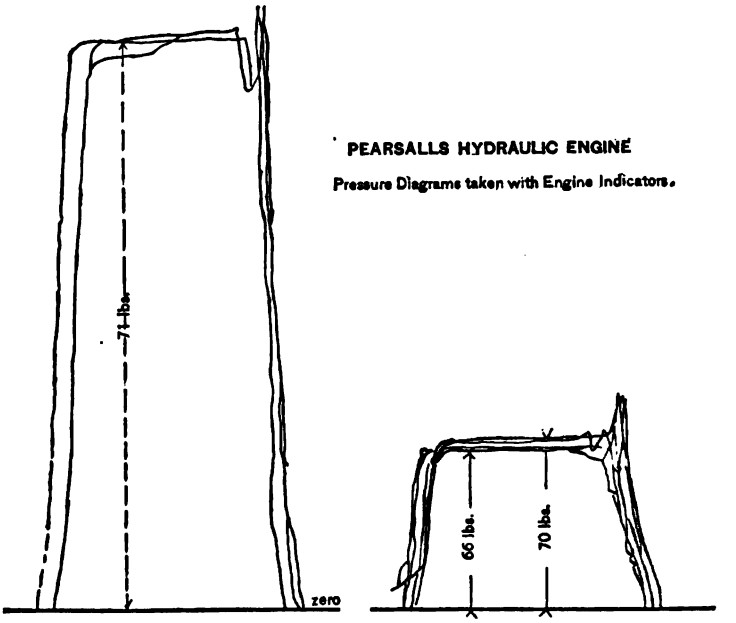
until it is *completely* closed, because, until the chamber F is filled with water up to the float H, the valve J remains open, giving free communication between chamber F and the atmosphere; consequently the air freely escapes from the chamber and the water freely rises in the chamber. This action takes place *during the closing* of the main valve C. The consequence is that no power is wasted in forcing water through the narrowing orifice of this valve. A second consequence is that there is no necessity to close this valve with great rapidity (which is necessary in hydraulic rams). As a matter of fact it is closed by a gradually retarded motion and so comes to rest without any concussion. When the water touches the float H, it closes the valve J, shutting off the passage for escape of air, and the pressure in the chamber then rises to the point at which the valves K open, and some of the water flows into the air vessel L, from which it of course is constantly flowing out through delivery-pipe M. A little air still remains in the chamber at the instant when valve J closes. This air is compressed and enters the air-vessel, and is used to drive the motor which actuates the main valve.

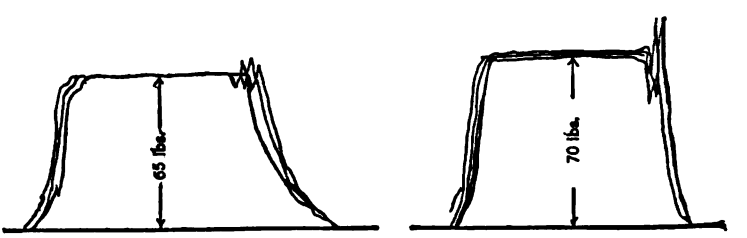
The column of water flowing in the flow-pipe is thus brought to rest entirely by the elastic resistance of air. When it has ceased to flow, the main valve is again opened, the water with which the chamber F is now full escapes, the air-valve J falls open, admitting atmospheric air, and the water again begins to flow and to escape through valve C.

The machine above described has been at work for several months with great success. Its action, as already stated, is exceedingly smooth and regular; no concussions or vibrations of any kind are perceptible; and the useful effect, as referred to more particularly below, is, it is believed, far greater than has ever been attained in the application of water-power to raising water. Fig. 2 shows pressure-diagrams which support these claims.

The engines which it is intended to construct for the placer-mine are similar in every respect to this one, but of larger size, and will supply 1000 miners' inches, or 10,000 gallons per minute. The existing engine forces only 200 gallons per minute under similar circumstances, but it will be readily apparent that its construction is equally applicable to a much larger size. Indeed, it is more suitable, the larger the size; for the greatest loss of work occurs through friction in the pipe, and this of course is greater, the smaller the pipe.

FIG. 2.


PEARSALLS HYDRAULIC ENGINE
Pressure Diagrams taken with Engine Indicators.
71 lbs.
zero
66 lbs.
70 lbs.


65 lbs.
70 lbs.

The size of the machine needed for the larger quantity will also be much less in proportion, on account of the fall used being greater. The quantity of water already dealt with is more than ten times that of the largest hydraulic rams in use, and is quite large enough to occasion difficulties if they were to be apprehended. The mass of water in motion is nearly four tons, but it comes to rest as quietly as can be desired.

The works to be constructed for the placer-mine consist of a short ditch about two miles long, which will be entirely made by plowing, and in alluvial soil which quickly becomes water-tight. In this distance a fall of 80 feet is obtained, and here hydraulic engines of the above description will be erected about half a mile up the gulch, the stream from the short ditch flowing through the pipe to the machine, and a proportionate part of the water being raised directly to the height desired.

Besides economy (which has been already referred to), the system has the following incidental advantages:

The water may be used with a high pressure even at the highest points of the placer, which is often difficult to arrange when using only gravity.

The pressure available for hydraulicking is not limited by the topographical features of the country, as is often the case in the ordinary system.

At the point where the machines are erected, a large extra supply of water is available besides that at high pressure, and will greatly help to keep the tail-flume clear.

As regards the efficiency with which the engine works, careful experiments have been made, gauging accurately the quantities of water used and delivered, and their respective heads. The head of supply was 17 feet. When head of delivery was 150 feet, the efficiency was 70 per cent., and when the head was 100 feet the efficiency was 72 per cent.

If this be compared with that of turbines and pumps combined it will be found to be very high; since, taking the efficiency of a turbine at 70 per cent. and a further loss of 20 per cent. on the pumps and gear, we do not get more than 50 per cent. net efficiency. In actual practice, I believe even this is not obtained.

A head of water varying from 10 feet to 40 feet has been used, and there appears to be no reason why these machines should not work equally well with heads varying from 2 feet to 100 feet.

The machine has been worked with water carrying in suspense large quantities of earth, gravel and stones, floating sticks and chips, and no obstruction to its working has been caused thereby.

A considerable variation of head and of quantity of supply, even during the working, does not affect the efficiency; and if these factors vary inversely at the same time, as they usually do, the quantity and pressure of water pumped remains the same.

The air-vessel is very easily charged with air to any required pressure by the direct action of the water.

Several other applications of the general design of this engine are being made, one of which I may briefly refer to, although the chief object of this paper is the use of the new engine in hydraulic mining.

By enlarging the size of the chamber F and causing the water to raise the float H at an earlier part of the stroke, the quantity of air forced into the air-vessel may be increased to any extent, even to the extent of absorbing the whole power of the column of water in compressing the air. In this way the engine becomes a very efficient air-compressor, and as the quantity of air left in the chamber to be compressed and forced into the air-vessel can be regulated to the greatest nicety by the adjustment of the position of the float H, this compression is effected without any waste of energy. In actual experiment as much as 90 per cent. efficiency has been obtained when compressing air.

It may be expected therefore, to be of great use in rock-mining and tunneling, for providing compressed air for rock-drills wherever water-power is obtainable.

Moreover it is not necessary that the water-power should be very near at hand, as the conveyance of compressed air in pipes for a considerable distance is quite practicable. Indeed it is proposed to utilize certain water-falls for miscellaneous motive-purposes at some distance in this way, and it is estimated that the power can be thus transmitted at considerably less cost than by electric conductors.

In conclusion I have only to regret that at the present moment the only engine constructed is so far from the meeting-place of the Institute, but I hope in the course of the year to have others working in the United States, and I shall be pleased to bring them to the notice of any member interested.

CONCENTRATING MAGNETITE WITH THE CONKLING JIG AT LYON MOUNTAIN, N. Y.

BY FERD. S. RUTTMANN, NEW YORK CITY.

(Boston Meeting, February, 1888.)

THERE is now in use at the mines of the Chateaugay Ore and Iron Co., at Lyon Mountain, New York, an ore-concentrating machine or jig, which, in view of the increasing attention given to the subject of concentrating lean ores of iron, seems worthy of notice.

This Conkling Jig, as it is called, is not entirely unknown, having been used at several places with more or less success; but as yet, I believe, no drawings of it or records of its work have found a place in print. Although the jig will be found to be merely an adaptation of old forms, there are several points which deserve attention, particularly the method of central discharge, of freeing the bearings from grit, and the rotation of the sieve or screen.

The ore-body at Lyon Mountain consists of a bed of rather compact, granular magnetite, occurring in a gneiss formation. The side walls of the ore-body are not sharply defined, the ore on either side becoming gradually poorer as the distance from the central point of the body increases. It is this lean ore, which, before the introduction of the jigs, was thrown away as waste, that is now concentrated. The material treated consequently consists of magnetite, intermixed with the component parts of gneiss, viz. quartz, felspar, hornblende, and mica. Grains of trap also form part of this material. These come from dykes of trap, which cross the ore-body at intervals.

The ore which is to be concentrated usually contains from 30 to 40 per cent. of metallic iron. It is first crushed to the size of two inches, or less, by two large Blake crushers. The material from these crushers passes by a shute lined with screens, having holes $\frac{3}{4}$ inches in diameter, to a second set of four crushers, set so as to close to one inch. The product from this set of crushers, together with that which passes through the screens in the shute, is then taken by an elevator to the top of the mill, and passed through a revolving screen, perforated with $\frac{1}{4}$ inch holes. The ore which passes *through* the screen goes to the jig-hoppers; that which passes *over* goes to a third

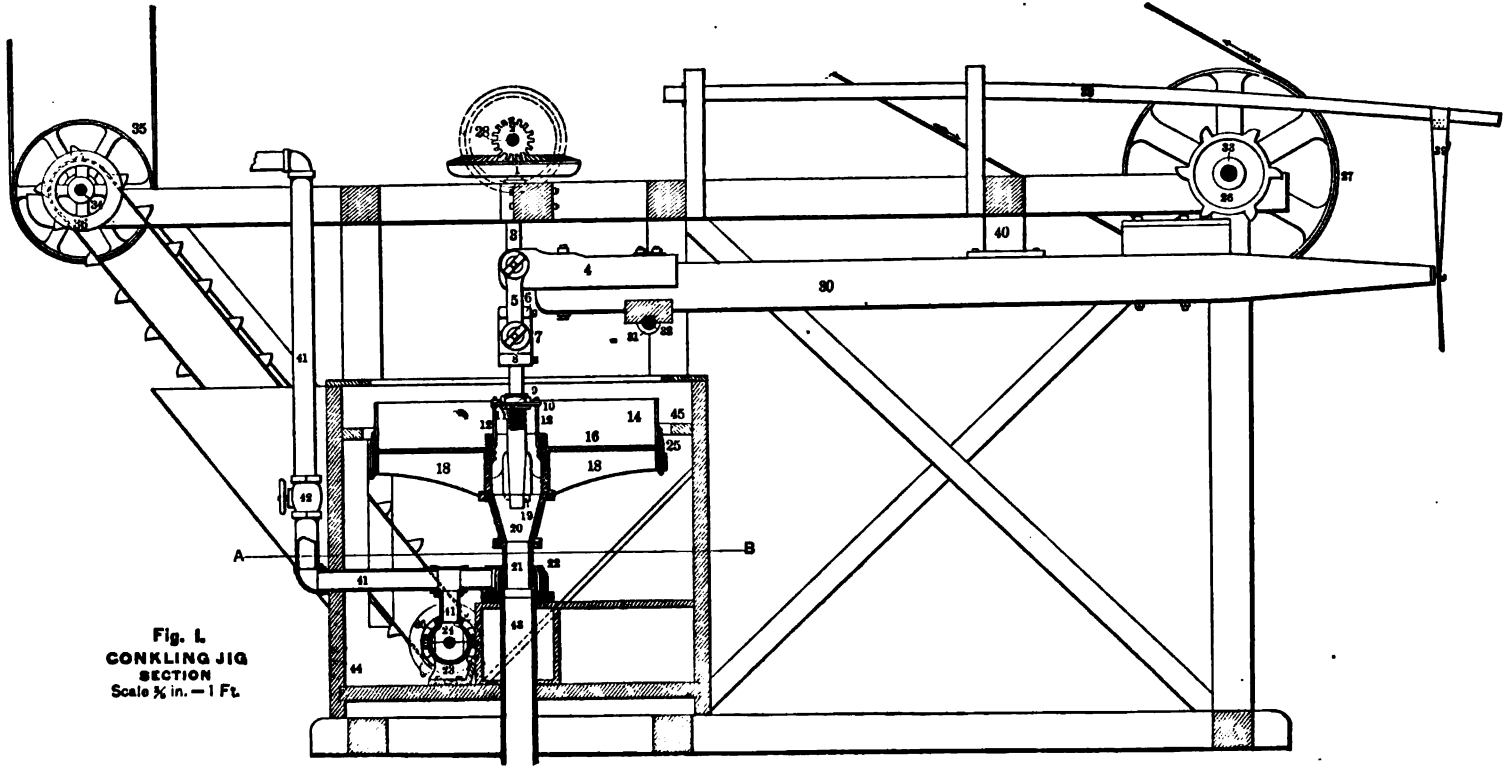


Fig. L
CONKLING JIG
SECTION
Scale ½ in. — 1 Ft.

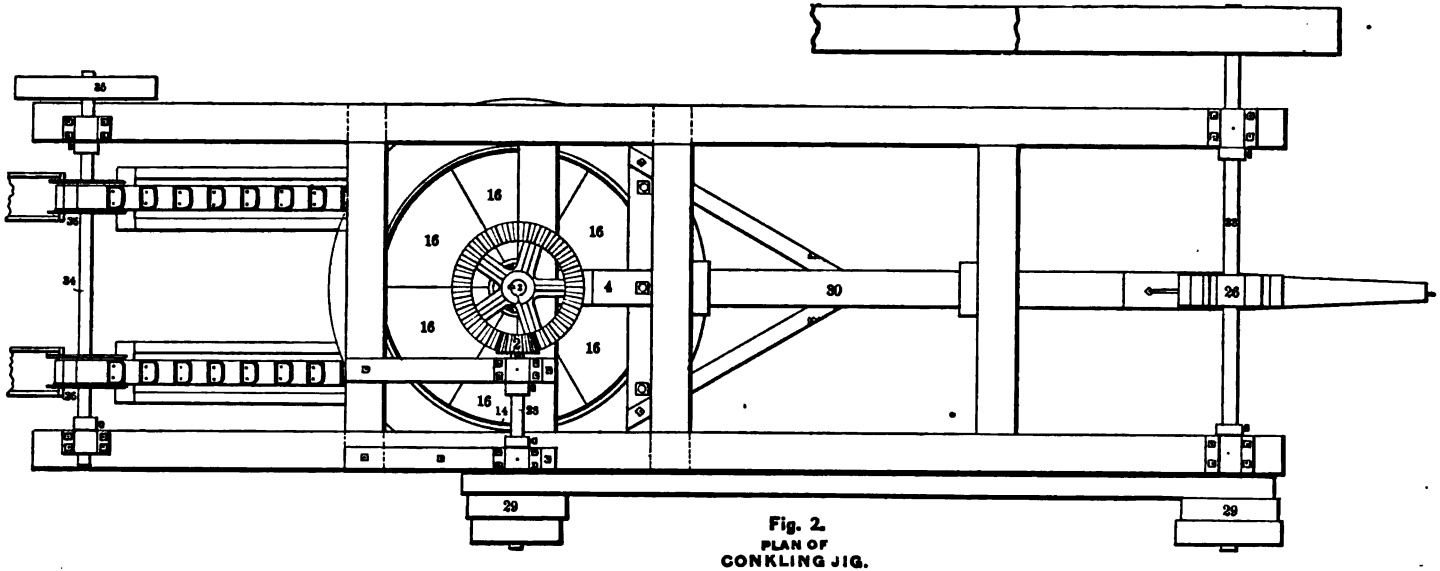


Fig. 2.
PLAN OF
CONKLING JIG.
Scale ¾ in. — 1 Ft

set of six crushers, set to ½-inch. These are of the Blake multiple-jaw pattern, having three jaws.

The product from these third crushers is again screened through a double revolving screen, having holes ½-inch in diameter. That which passes through goes to hoppers; what passes over goes to a fourth set of crushers. These are of the Blake multiple-jaw pattern, having six jaws, which are set so as to close to ½-inch. The product from these last crushers is again screened, and the grains which are still too large pass again through the final crushers. All the material which passes through the ½-inch holes goes to the jigs without further sizing. The ore is crushed dry, and goes to the jigs in that state.

The machine, as shown in the accompanying drawings, may be divided into four essential parts: 1. the frame; 2. the tub; 3. the jig; 4. the elevators. For the better understanding of the drawings, the following index of the names of the parts is appended, the numbers referring to corresponding numbers in Figs. 1 and 2.

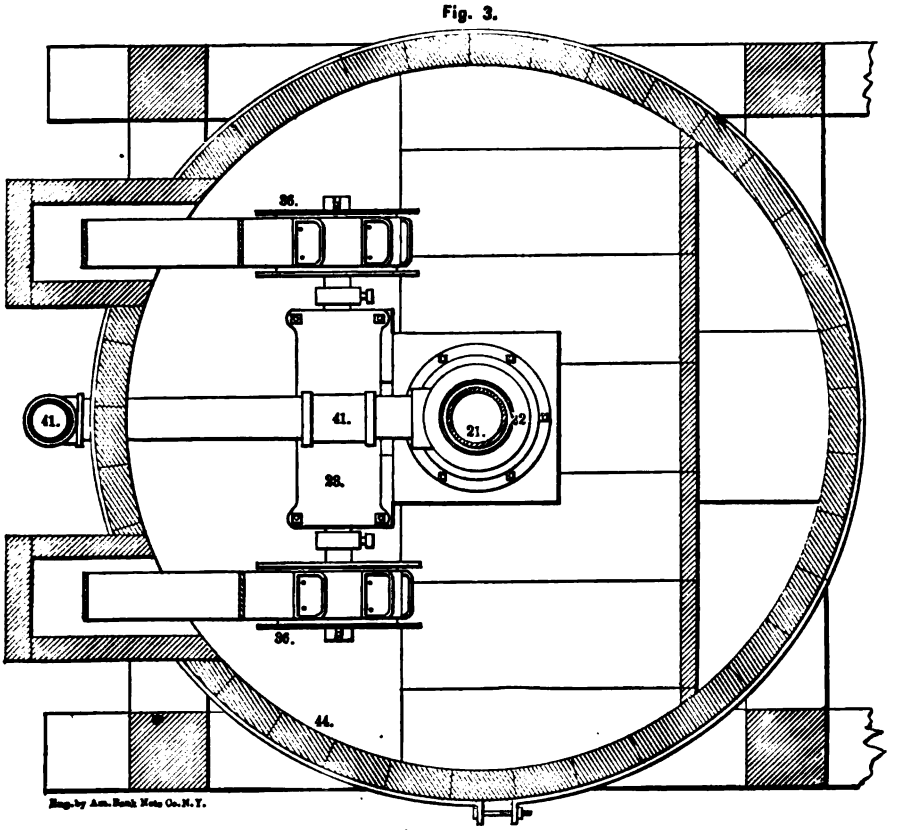
INDEX.

1. Bevel-wheel.
2. Pinion.
3. Upright shaft.
4. Shoe.
5. Link.
6. Upper collar.
7. Trunnion-piece.
8. Lower collar.
9. Outside nut.
10. Flange.
11. Inside nut.
12. Standard.
14. Hoop.
16. Screen-plates.
18. Spider.
19. Key.
20. Cone.
21. Water-sleeve.
22. Water-box.
23. Lower elevator-box.
24. Lower elevator-shaft.
25. Band-arms.
26. Cam-wheel.
27. 36-inch pulley.
28. Pinion-shaft.
29. Cone-pulleys.
30. Lever-beam.
31. Yokes.
32. Lever-shaft.
33. Driving-shaft.
34. Top elevator-shaft.
35. 24-inch pulley.
36. Flange-pulleys.
37. Cone-pulleys.
38. Spring-pole.
39. Strap.
40. Bumper.
41. 3 inch water-pipe.
42. Regulating-valve.
43. Outlet-pipe to tail-race.
44. Tub.
45. Splash-rim.

The Frame.—The frame, which is shown in vertical section and elevation in Fig. 1, in plan in Fig. 2, and in horizontal section in

Fig 3, is made of squared spruce or hemlock timber, 6½ x 6½ inches, and braced by cross-bracing, 3 x 4 inches, at the sides and ends. The upper string-pieces project 4 feet 5 inches beyond the first pair of uprights. The sills are 13 feet 8½ inches long and project 6½ inches beyond the front and rear uprights. The second pair of uprights is placed 3 feet 9 inches behind the first pair. The frame

Fig. 3.

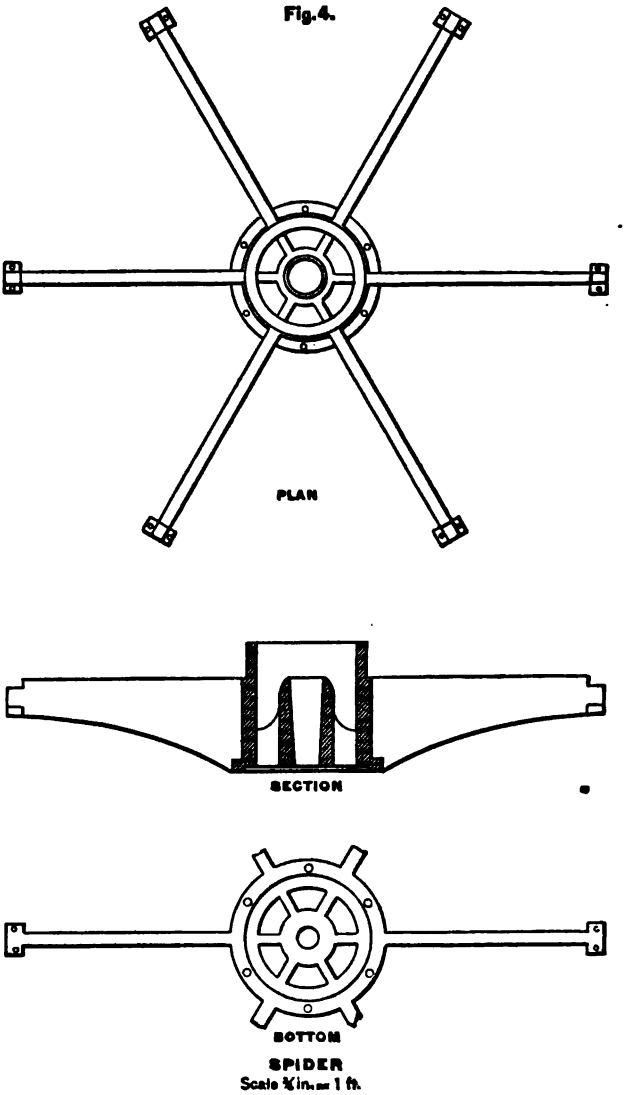


SECTION, (HORIZONTAL) ON LINE A-B FIG. 1
Scale ¾ in. = 1 ft.

is 4 feet 1 inch wide inside. There are an upper and a lower-cross piece at each pair of uprights except the rear, where the cross-piece is placed 3 feet 8 inches from the rear end of the string-piece. To the lower side of this cross-piece is attached the bumper for the lever-beam. The main driving-shaft is placed directly over the rear uprights; and 9 inches from the forward end of the string-pieces is placed the shaft for the elevator-pulleys. Between the first and

second cross-pieces is a cross-beam which carries the journal for the upright or jig-shaft. The front edge of this timber is set even with the center-line of the jig. From this timber to the forward cross-piece extend two timbers, 4 x 6 inches, as shown in Fig. 2, which

Fig. 4.

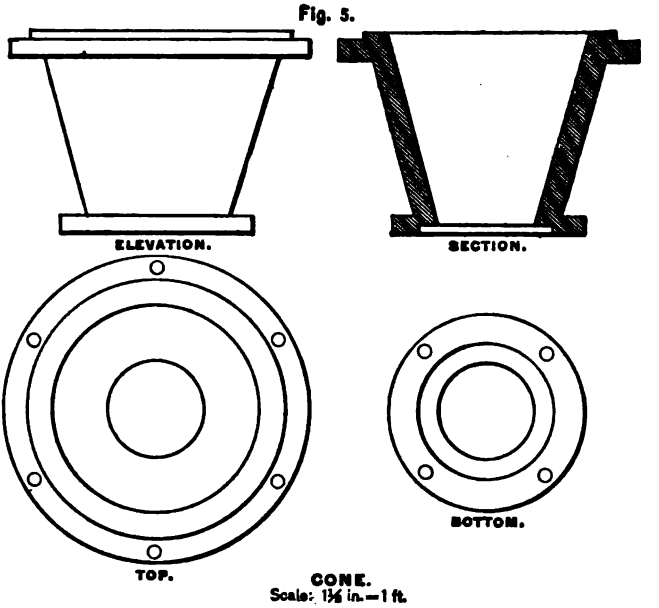


carry the boxes for the pinion shaft. To the second and rear cross-pieces are bolted pieces 3 x 9 inches, and 2 feet high, which carry the spring-pole. On the front of the second pair of uprights 1 foot 6½

inches (to center) below the string-piece, are the journals which carry the lever-shaft. The whole is framed with mortise- and tenon-joints, and tied together with 1 inch bolts.

The Tub.—The tub is made of yellow pine 4 feet 11 inches inside diameter. The staves are 4 feet 8 inches high, 5 inches wide and 2 inches thick, rabetted 1 inch deep, 3 inches from the lower edge, to receive the bottom. The staves are not dowelled. A door in the side having bevelled edges gives access to the interior. Four iron bands provided with tighteners binds the whole together. A hole near the bottom fitted with a plug serves to empty the tub for the

Fig. 5.



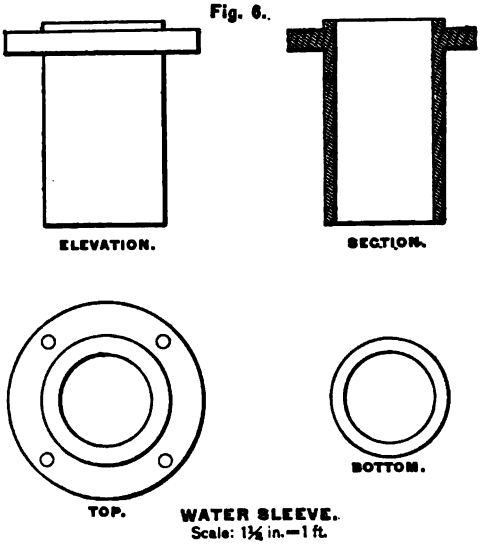
CONE.
Scale: 1½ in.—1 ft.

purpose of examination or repairs. The tub is set between the first and second pair of uprights, as shown in the drawings. Two troughs to allow for the passage of the elevator belt and buckets into the tub are let into the front side.

The Jig.—The jig consists essentially of a circular sieve, suspended from one end of a lever, being moved up and down by a cam striking the opposite end of the lever. The concentrates pass through the sieve to the bottom of the tub; the tails pass out by means of an annular opening around the jig shaft. The general arrangement is shown in Fig. 1. The parts are shown in detail in Figs 4, 5, 6, 7, 8, 9, 10, 11 and 12.

The spider (Fig. 4) is made in one piece of cast-iron, with a taper bore to receive the jig-shaft which is keyed into it. It is also supported by the standards from the flange, which may be moved by the upper and lower nuts. A sheet-iron hoop, 12 inches high, is bent around the spider, and fastened by the holding-down bands which are riveted to the rim, pass through the holes in the end of the arms, and are fastened below with nuts. The screen-plates rest on the arms of the spider and are held in place by U-bolts passing under the arms and through the holes in the screens. The screen-plates are $\frac{1}{2}$ inch thick, made of cast-iron, in segments of $\frac{1}{6}$ of a circle; the holes are $\frac{5}{16}$ inch in diameter on top and $\frac{7}{16}$ inch below.

Fig. 6.



WATER SLEEVE.
Scale: 1½ in.=1 ft.

Beneath and bolted to the spider is the cone (Figs. 5 and 20, Fig. 1) under that is the water-sleeve (Figs. 6 and 21, Fig. 1) which slides up and down in the water-box (Figs. 7 and 22, Fig. 1). This water-box, and also the lower elevator-box (Figs. 9 and 23, Fig. 1) are the inventions of Mr. William Hodson, Superintendent of the mill at Lyon Mountain. All the water which is to be used in jigging passes through these two boxes, and flowing out through the annular openings, keeps the bearings free from grit. Previous to the introduction of these improvements, the water-sleeve would wear out in a very short time. The water, under pressure of 8 feet head, enters through the 3-inch pipe (41, Fig. 1) provided with a valve (42) to regulate the quantity.

nd of the lever-beam
7, Fig. 1) is kept in
re provided with set
nect the jig with the
rough the horizontal
ed; the shaft moves
nes in which fit keys
en by belt from the
transmit the rotary
ge the speed. The
vided with six cams,
t 8 inches wide, pass-
l). The cam-wheel
260 jars per minute

SECTION.

the jig up and down
down-motion. The

own in detail in Fig.
The box is bolted
or-pulleys (Fig. 12)
water which passes
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om the hoppers to the
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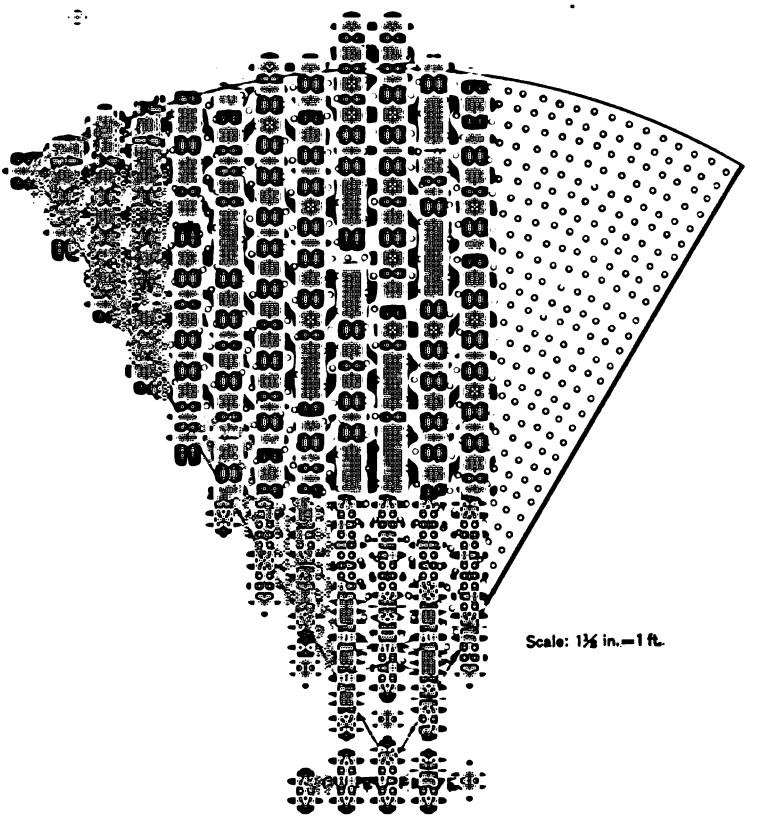

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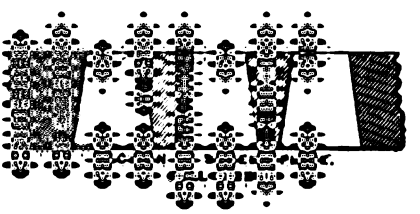
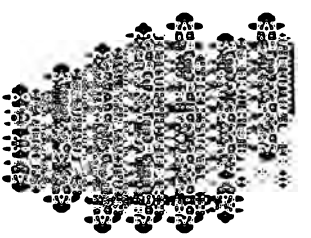
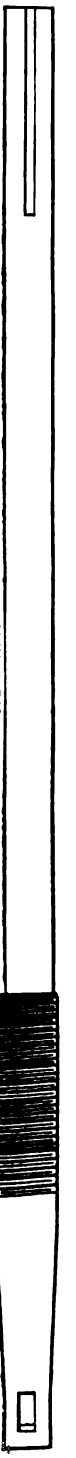
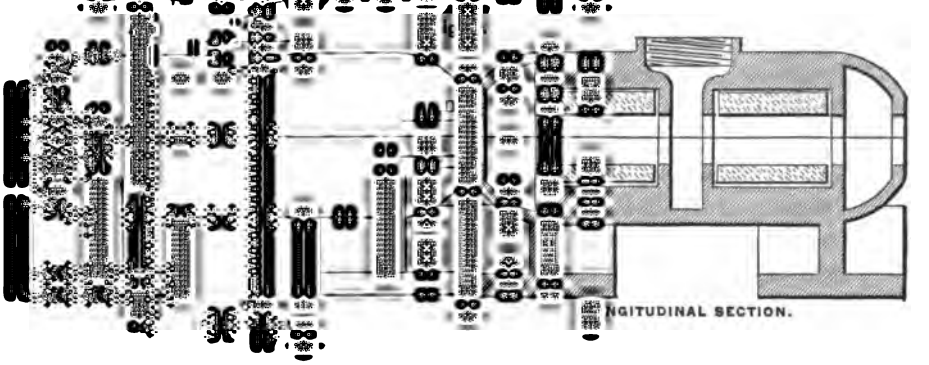




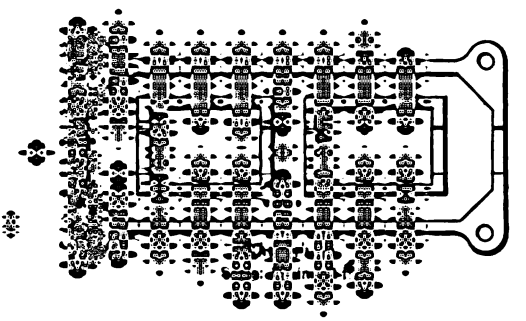


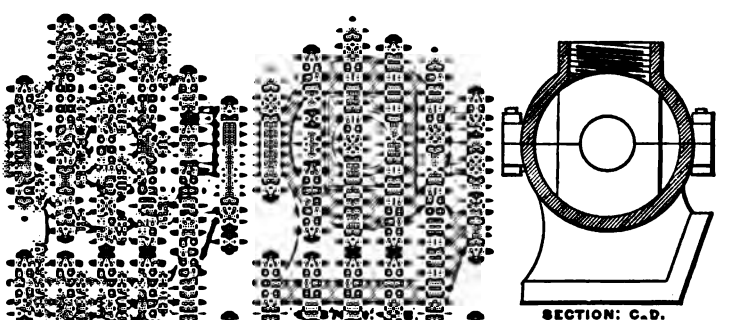
JIG-SHAFT.

[illegible] hected with the lever- [illegible] the jig started. The



NGITUDINAL SECTION.





SECTION: C.D.

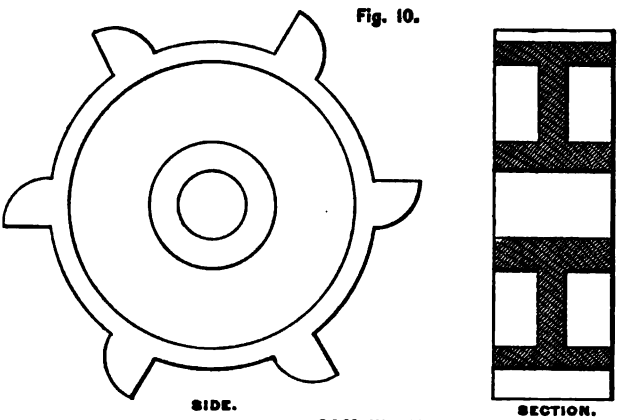
[illegible] ALLEYS.

[illegible] es and over the collar [illegible] l-race; the ore settles

through the screen, is collected at the bottom of the tub, and thence raised by the elevators to bins. The rotation of the jig produces an equal distribution of the crushed ore on the screen-plates, and also forces the particles to traverse a path longer than the radius of the sieve. The crushed ore is allowed to fall on the screen as near the outer periphery as possible.

The jig has a capacity of treating 5 tons of ore per hour, requiring 135 gallons of water per minute, or 1620 gallons per ton treated. One man or boy is sufficient to attend to two jigs, his duty being to see that the jig is properly supplied with material to secure a uniform depth on the screen; that the water is flowing in the proper quantity (a depth of $\frac{1}{4}$ to $\frac{3}{8}$ of an inch over the collar being suffi-

Fig. 10.



CAM-WHEEL.
Scale: 1½ in.=1 ft.

cient), and that the concentrates are free from gangue. If there is too much gangue in the product the attendant lets out a hole in the strap from the spring-pole to the lever-beam, to make the shock less violent.

An examination of the bed on the screens during the process of jigging showed the ore lying just above the screen, the greatest depth and largest pieces being near the edge of the screen, gradually thinning out and becoming finer toward the center of the jig, the gangue increasing in depth toward the central point of discharge.

An analysis of samples, taken on October 10th, 1887, gave the following results, the samples being analyzed for iron only:

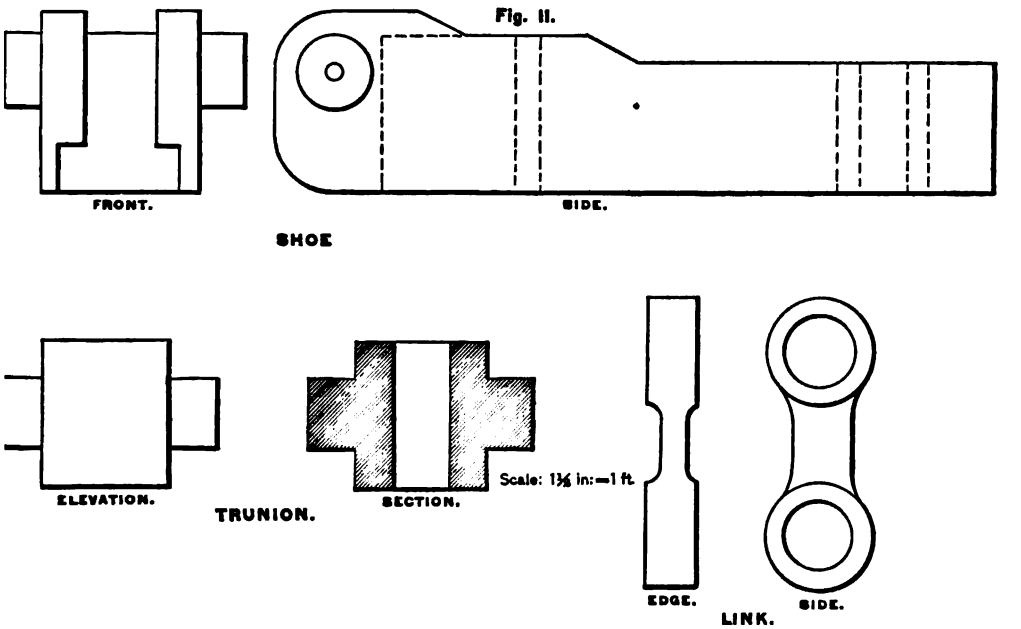
	Metallic Iron. Per Cent.
"Stock," or material treated,	43.56
Concentrates,	66.90
Tailings,	22.91

which shows a saving of only 47.4 per cent. of the iron contained in the stock.

The chief impurity contained in the concentrates appears to be silica in fine grains. An examination of the tailings shows the lost ore principally in two forms: very fine ore or "slimes;" and small particles of ore imbedded in the larger pieces of gangue. The causes of this loss may be summed up as follows:

1. The material is delivered to the jigs dry; hence the fine ore, almost in the condition of dust, is carried over into the tail-race on the surface of the water.

Fig. 11.



2. There is no classification according to size; and the ore, being friable, is crushed finer than the gangue. Hence, without sizing, there is not sufficient difference in the falling-time between the fine ore and the coarse gangue to produce a good separation, and fine ore goes over with the gangue into the tailings. A larger saving would probably be effected by a sizing into at least two classes; still, this would not save the ore which is lost through being imbedded in the grains of gangue.

3. The loss last noted is due to the fact that the material is not crushed fine enough to free the ore from the gangue. The remedy

is, of course, a finer crushing; but this would produce a larger percentage of fine ore or slimes, which is not desirable.

A better arrangement for treating this class of ores with this jig would probably be as follows:

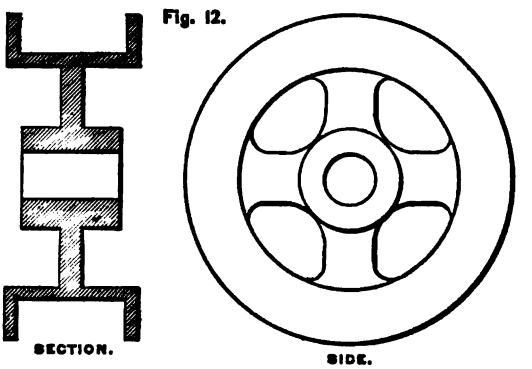
1. A wet crushing with stamps, say to the size of $\frac{1}{4}$ inch. The Ball steam stamp would probably work well in this connection, and produce a smaller percentage of fine ore than dry crushing.

2. A first jigging on a set of jigs arranged for this size, practically like those at Lyon Mountain.

3. A screening of the tailings from first jigging by a screen with $\frac{1}{8}$- or $\frac{8}{82}$-inch holes. The material above that size to be recrushed, either by being sent back to first stamps or by a second stamp.

4. A second jigging of the tailings from the first jigging which pass through the $\frac{1}{8}$- or $\frac{8}{82}$-inch screen.

Fig. 12.



ELEVATOR PULLEY.
Scale, 1½ in.= 1 ft.

If it may be supposed that the second jigging will be as perfect as the first this arrangement will probably result in the saving of an additional 20 or 25 per cent. of the iron contained in the stock.

The economical practicability of this arrangement may be questioned on the ground of expense, taking into consideration the comparatively low price of the product obtained. As I have no means of knowing either the cost of treating per ton of material, or the price obtained for the product, I must leave this point in doubt.

There seems to be considerable room for improvement in the arrangement of the jig. As it now is it is a somewhat clumsy arrangement, and occupies a space 10 x 24 feet (allowing 2 feet of working space around the frame). This could easily be reduced to 10 x 10 feet by doing away with the lever, spring-pole, etc., and giving the

motion to the jig-shaft by means of a cam working on a horizontal shaft just back of the jig-shaft. The spring-pole could be replaced by a coil-spring on the vertical shaft, with an adjustable nut to regulate the jar. A suitable bumper could be arranged either above or below the bevel-gear wheel. The action of such an arrangement would be similar to that now produced, with the advantage of economy of space and material.

SOME THOUGHTS AND SUGGESTIONS ON TECHNICAL EDUCATION.

BY T. EGLESTON, PH.D., NEW YORK CITY.

(Presidential Address at the Boston Meeting, February, 1888.)

FOR a great part of the progress of the world we are indebted to the works of engineers. It is to them that we owe our means of rapid transportation, our canals, our railroads, our bridges, many of our automatic machines, and most of those appliances which, by facilitating or decreasing the amount of human labor to be expended and by utilizing the great forces of nature, have made the present progress of the world possible. Their education is a matter of momentous interest to the whole civilized world. I have, therefore, thought that some discussion of the theory upon which their education should be based would be of interest to this body, and have ventured a few reflections and suggestions upon the subject.

The word education, in its original signification, meant the drawing out of a man what there was in him. In our common schools the practical work of education consists in putting into the brain a larger number of facts than can be retained, which are consequently more or less understood and digested. It ordinarily consists simply of teaching a round of abstractions or facts which when not assimilated are not only useless to the individual, but generate careless habits of thought and make more or less untruthful, unfaithful or unsuccessful men. In most of our schools a procrustean system is adopted; all are made to advance upon one general plan, the quick being kept back and the slow dragged on at a rate too great for their power of comprehension. Until within a very few years it has been a theory in common school education that the brain alone should be developed, and that little or no attention need be paid to the train-

ing of the other parts of the body, so that the eye, ear and hand which, for practical life, are the most important parts of the body, are left, we might say, entirely uneducated. The child is constantly told that to make his manhood successful he must *do* something, yet no training is given to any part of the body with which action is possible. So far as the rest of his faculties are concerned, his brain is an abnormal development. On the one hand he is taught to "live by his wits," and on the other that a person who does so becomes a byword and a reproach. But true education should bring about the harmonious growth of the whole man, and not a one-sided development of one or two of his faculties.

With this system there has been for some years a very general distrust and an effort has been made to break away from it and replace it by some other method. That it has not fulfilled what was expected of it is patent to almost every one, and should we judge of it only by results, as we should judge anything else, it would seem at once that it was, to say the least, not satisfactory. It has been so long the habit to declare that our public school system of education is the foundation of all that is good in the country that it is difficult to look at it in any other way. We have only, however, to compare the results of such training with those of our own profession to see that it must be very defective. I remember in the year 1860 having heard my father, then 60 years of age, say of the men who started in business with him as a boy, that 90 per cent. had failed at least once, and that by far the larger part of them had never succeeded in obtaining even a competency. I have since heard the estimate of failures placed much higher. These men had all been educated under the theory then most common in the schools. No such statement could be predicated of the engineers who started in life at the same time, nor of the education which was given in the engineering schools. Any such percentage of failure in their graduates would have wiped them out of existence and consigned them to oblivion long ago. It is remarkable, in view of the fact of the constant failures in the common school education and its manifest unfitness, that attention should not sooner have been called to the adoption of the methods used in engineering schools, and that it should have required so many years to turn attention to the fact that there was something decidedly lacking in the inception and execution of our general school system, and more especially that it was training our youth in such a one-sided manner that brain development only was regarded as the legitimate object of education.

Bacon has said that education is the cultivation of a just and legitimate familiarity between men and things. Judged by this standard it must be said that our common school education is lacking in some of the essentials which bring about this familiarity. Facts in themselves are of no value; they are only important when they are applied. Mere thought in itself is of no use; it only commences to be of importance when it is put into words; but it appears in its full power when action follows the thought. The real value of education depends not upon acquired knowledge, but upon the uses that can be made of it. Its value to the individual depends solely upon his ability to apply it and upon the habits of mind which it creates. Simply knowing how to think is of no value if there has been connected with it no education of those faculties which make it possible to put thought into action. The world would never be influenced by mere thought. We might say that thought, as such, is of no possible use. The same might be said of unguided action; but aided, the one by the other, they make the power which moves the world. It is this combined power of thought, followed by action, that the engineer constantly has to practice, and which the student ought to learn in the schools. It is this association which makes the superiority of the men who have, and the inferiority of those who have not, been trained in it. If the receptive faculties only are to be cultivated it is very doubtful whether any better discipline can be obtained by the reception of one set of facts or ideas than by another, for the opinions of to-day are often found to be erroneous to-morrow; but in the application of principles through the eye, the ear, and the hand there are so many mental qualities brought into play that the result is always on the side of science, which in its application is another term for progress. It is this which characterizes the education of engineers and distinguishes it from other systems. For many years they occupied this ground almost alone.

How little can be said on the other hand for our common school education. We constantly hear of over-taxed brains or health injured by foul-air and want of proper exercise, or children unfitted for the station in which they are born and not fitted for any other, and, as the result, of dissatisfied and querulous men and women, who despise the poor work of their untrained hands and have not been sufficiently brain-trained to turn what little education they have into remunerative work. We are looking about for a remedy, and education in this century is consequently in a transition period. We

are justly dissatisfied with the old system, which has been weighed in the balance and found wanting, and must therefore be for our common schools greatly modified or entirely discarded and some other put in its place; and we are casting about to find out what that other shall be. Why should we not adopt something like the method which has been found so successful in our education of engineers?

In the middle ages learning was confined to the monasteries, and consisted for the most part in the ability to take part in metaphysical discussions on subjects of but little importance and in logical hair-splittings of no practical use. When the monasteries lost their power the medical profession took their place, and to it we owe much; but charlatanism was often the only result of its observations, though it produced the alchemist and his successor, the chemist, to both of whom the world is greatly indebted. Their place has been almost wholly supplied by engineers or men who apply what they know to some practical purpose, to whom we look for the establishment of the great principles which make both the progress of the world and the advances in science and its applications. It does not appear that the law ever held a similar position. It seems always to have been busy in unmaking what its predecessors had done, and in making something new to take its place, which was in its turn undone. The monasteries represented the metaphysical side of education; medicine, mere observation and collection of facts without their application; and engineering, the age of investigation and the application of facts to what is useful to man. Consequently engineering represents progress both in science and art, and unites all that is best in both of the former systems. The world has shown its appreciation of this method by demanding the applications of things. To say that a thing is of no use, *i. e.*, that it has no application, is equivalent to its entire condemnation.

Formerly the application of facts was made largely by the men who gained their knowledge in some very limited field in which they made use of them, and they were called "practical men." The man from the schools was a theorist. The practical man was one who had learned to do some one thing, in distinction to the schoolman, who first theorized how anything should be done and then undertook to do it. In that sense the demand for the practical man, which was urgent a few years ago, has almost ceased, for it has been found that the graduate of the school, with his manual and mental training and knowledge of how things ought to be done and where to look for what he does not know, easily replaces the practical man when he

has been but a short time in the profession, and far exceeds him in ability when he has had a small amount of experience, since he reasons as well as acts, while the practical man acts without reasoning and has only his own limited experience for his guide. The present demand is therefore for such men as can deal with both sides of the problem. To the practical man, a new field is a new creation just as much as if it had never been in existence before, while to the man of the schools it is only another phase of a problem with which he is more or less familiar, or if he is not familiar with it at all, the mental discipline which he has acquired fits him fully for mastering it. I have, and always have had, a great esteem for the practical man in his proper place and rightly understood. He has acquired his knowledge by rule of thumb, and knows only the experience that he has gained in particular cases; but he has usually learned by this experience so well and has cultivated certain of his powers to such an extent that he is generally only at fault where his experience fails. If he has had a number of special cases and been successful in them he may be successful again; but his knowledge is almost always empirical, his experience confused and confined within very narrow limits, but when with his practice, his powers of observation have been cultivated and he has learned to reason he rebels against the name of practical and becomes at least the skilful man, and when, as is sometimes the case, he carefully studies principles, he becomes an engineer. I have nothing to say of the abuse of this word "practical" by men who use it to cover up ignorance.

The education of engineers, however, has been usually better conducted. They have not only had the receptive faculties cultivated, but the eye, the ear and the hand have been so trained that a failure with them is uncommon; and it is to this distinction between the education of engineers and ordinary education that I wish particularly to call attention. Until within a very few years France has led the world towards solving the problem of technical education. The reason for this was not only on account of the proper system and the greater advantages which were given to the students, but was also due to the fact that a system of selection by examination was adopted. Commencing at a very early age, the children all over the country were sorted and resorted until when they arrived at sixteen they were definitely classified into those adapted for the various professions and were then sent to the different technical schools supported by the government to be trained and after graduation were again re-selected, according to

their capacity or ability for the different branches of each profession. Such a system raised education in France to the highest point it has ever reached and produced some of the most profound thinkers and investigators that have ever been known. In Germany great stress has always been laid on what they call their "*practicum.*" Their success consisted in training men in the schools in what they would be called upon to do as soon as they left them for practical life. It is to the combination of these two systems, a careful selection by examination and the practical training both in and out of the schools, that the most of our success in educating engineers has been owing.

These methods, which we are constantly amplifying and improving, we call scientific. They have grown with the results of experience, but to most men science, which really means only knowledge, is regarded as just as fixed and positive as mathematics, while in fact, as science is only knowledge, its value is in reality only relative. We are constantly told, "these are the deductions of science," as if they were perfectly mathematical and certain; whereas the best science of the last century would be dense ignorance in this, and what is scientific and true to-day, with the knowledge we have, may become to-morrow, when further facts are discovered, entirely untrue. Science is really only a round of invention and discovery. With the knowledge that we have we reason and conclude, until we find by experience that our reasonings are false, since the results are not true, and then, with the facts that we have at our disposition, we are obliged to take up some other theory. It is this high endeavor to ascertain what is truth and to apply the principles which are involved in it, which make the active searcher after truth, and ought to develop the character and make high toned men, and it is to this constant adaptation of our technical school system to the needs of the day that its superiority is owing. Could the same be said of our common schools we should have less fault to find with them and there would be fewer failures among their graduates.

All education should tend to the development of the reasoning faculties in such a direction and in such a way as to enable a man rapidly to determine what truth is, and then if the eye, the ear and the hand have been properly trained, knowledge and experience will give him the power to apply it. As such an attainment is a matter of the highest importance I propose to call your attention to what I consider to be some of the defects and advantages in technical education, in the hope that from the comparison some suggestion may be made towards an amendment in its system and lead to the adop-

tion of a different method in our common schools. Our technical schools are designed to train engineers. But what is an engineer? The people think of him as an engine-driver. The dictionary says he builds engines of war and steam-engines, or constructs roads and bridges. In England he is a constructor of almost anything. There is need of a better definition in these days, when we have military, civil, mechanical, mining, chemical, sanitary, and electrical engineers. These men originate and design what is adapted to new and constantly varying conditions, apply old principles to the solution of new problems, and keep the world constantly advancing. In those parts of the world where there are no engineers, civilization either retrogrades or is at least stationary. There are so many special traits of character which distinguish engineers as a class, that it is most interesting to study how they have been formed and especially to observe how the habits of accuracy and love of truth have been and are to be developed; and I have thought it would be appropriate to discuss how engineers should be educated.

In most of our schools, if not in all, the requirements for admission are too low. The student enters so badly equipped that from a year to a year and a half is expended in teaching him the mathematics and other preliminary studies, which he must use in the later years of the course, so that any application of these studies must be confined to within a little less than half of the entire school-training. The justification of this course has always been in the assertion that the preparatory schools cannot be made to raise their standard sufficiently high. Yet during the last twenty years their standards have been raised; and it is to be hoped that at least a whole year of instruction in the technical schools may in future be gained by demanding for admission to them, mathematics up to and perhaps including the calculus, so that the applications only of mathematics may be taught in these schools. The same might be said of some of the other preliminary studies. If this were done gradually, I think there would be no serious difficulty in increasing the value of technical education without lengthening the course of study and practically securing what would now be equivalent to a five years' course by throwing out into the requirements for admission a whole year of the present instruction. This would permit, at least in the last year of the course, the practical application of the principles taught, and the introduction at that time of a commencement of actual professional field-work, and especially the more extended carrying out of the principles of engineering as ap-

plied to constructions executed on a large scale, and the introduction of the practice of the principles of design, so as to teach the student what can or cannot be made in wood or metals and the general application of these principles to economic construction.

A knowledge of French and German is absolutely essential to success in any professional career; and as this knowledge is wanted, not only in the school but out of it, it seems proper that it should rather be demanded as an entrance-requirement than that instruction should be given in these subjects in the school. For many years, in the institution with which I am connected, instruction in French and German was given as a part of the school-course, but this method finally had to be abandoned and it is now placed in the requirements for admission. It was found almost impossible in the early days of that school to get the preparatory schools to promise beforehand to undertake instruction in this department, but since we have required it there seems to be little or no trouble in inducing them to do so and a great deal of time is thus gained which is now devoted to other important subjects. It seems much more rational that it should be required for admission than that it should be taught in the school, for a very large part of the investigation that is being made on professional subjects and a very large number of valuable communications which are being presented to the scientific societies and the periodicals are published in those languages; and as there is constant occasion to refer to them, the student is absolutely at a loss in this matter unless he can consult easily and without much expenditure of time such periodicals, which become of course a terra incognita, unless he is sufficiently familiar with these languages to read current literature fluently. The change made in this respect has been to us most beneficial and I think would be likely to prove so in other institutions if it was adopted. The teaching of languages also seems to be rather foreign to the objects of a technical school. At any rate we have found the greatest advantage in devoting the time that was formerly given to it, to strictly professional studies and demanding that the students entering should have a sufficient familiarity with these languages to be able to read current literature. Such studies as history, geography, logic, rhetoric and English composition are quite as necessary to engineers as to other men and where collegiate education cannot be had, should take, as far as possible, the place of such instruction, and in order that the young engineer shall have something equivalent to a collegiate course, these should also be required for admission. Composition, which means the art of expressing thought, ought to be not only

required as an entrance study, but should be fostered in all the written exercises of the school. A drawing speaks for itself if made in sufficient detail. It is made in the universal language which has within itself all the logic as well as the rhetoric of the schools, but composition gives the means of expressing concisely and logically in words ideas which thousands, who do not understand that universal language, can comprehend. It also gives the power of setting one's own thoughts logically and tersely in order, and is one of the best means of ascertaining for the engineer both what he does think as well as what he ought to think, and is often a better means of convincing himself than of persuading others. A plain, terse style of composition is a mighty weapon of war which engineers, of all men, use to their own hurt if they do not use it right. In any case such training ought, if possible, to be acquired before entering the technical school as will be to a great extent equivalent to a university education. It would be impossible, of course, to obtain the whole of it, but there is great need that more attention should be paid to this subject in our technical schools, in order that we may have cultured as well as educated men in the profession. The remarkable unanimity with which all those who in 1876 joined in the discussion on technical education before the Institute at Philadelphia, demanded the greatest possible culture in the applicant for admission to our technical schools, showed how necessary in the opinion of every engineer, long in the profession, this previous training and its consequent culture was thought to be, for successful professional practice. There is a real lack in most of our technical schools in that moral science is neither taught nor required for admission. It has been left out in both places partly through inadvertence and partly because in the early days of the establishment of technical schools in this country most of the students came from the colleges, and therefore had this training; but now that the tendency is to replace the college by the technical course, attention ought to be given by our educators to the fact that certain studies of the university which promoted especially certain moral qualities are now entirely neglected. They are not taught in the preparatory schools and are left out of the technical course. It may be said that these moral qualities are produced by the constant search after truth, which the engineer must of necessity make in order to be assured of success in the practice of his profession; to which it must be replied that moral qualities are usually the result of education and that where the principles of morality are not taught directly the indirect

teaching of the laboratory and workshop and the endeavor to apply to actual practice the great principles which result from a constant study of the forces in nature do not of necessity create a thorough understanding of the basis of those moral principles which are necessary to the well-being of a man. Some provision ought therefore to be made, either in the school or in the requirements for admission, for the teaching of the general principles of morality.

One of the chief things which force themselves upon the attention of the instructor in the scientific school, is the apparent disadvantage which the college men appear, at first, to have in comparison with those who have only received what training they have in the scientific school, and it seems sometimes as if the advantage of a college education was very much overrated. It has, however, been the constant experience of the institution with which I have been connected that the culture which the college man has, while it does not show in the preliminary years in which mathematics are almost exclusively taught, makes itself felt the moment the applications of principles come to be made, and shows itself constantly in the practice of after-life.

But the old university education was only a preparatory training for the "professional schools," as the schools of law, medicine and theology were formerly called. At first the technical school occupied a position similar to the "professional school" with excellent results. No university man who has graduated at a technical school, and has employed his time at the university well, can call it wasted. The profession has still every reason to be proud of university men who have supplemented their education by a course in a technical school. It would be greatly to the advantage of such technical school if entrance into it could be preceded by a university training; but as we are every year diverging more and more from this practice, and as now such double training is not generally practicable, there is gradually ceasing to be any intermediate between the common and the technical school. The object of the technical training is now to fit one for practical life at an age younger than that at which men usually graduate at the university. The university course has been supplanted by the more or less perfect instruction in the common school, and the technical school for the engineer now supplants the university, and he thus enters upon his technical school training without any such preparation as the professional man, as he was formerly called, had. How shall we supply to the applicant for admission to our technical schools the training which the young man entering the

schools of law, medicine, and theology from the university is supposed to have? The answer to this question is plain, and it is the only one that can be given. It is by requiring for admission to the technical schools a high standard of attainment and such studies as shall be fully equivalent to the engineer for the loss of the university training. Among these requirements should be included the elements of rhetoric, logic, moral and intellectual philosophy, political economy, history, English composition, and at least two of the modern languages. It is hardly to be supposed that all of these could be taught in the common schools to the same extent as in the university, but their elements could be. There would, then, be much less danger to the engineer from the want of thorough acquaintance with these studies if he was drilled in the elements, since it is his peculiar province to know how to find any information for which he has need. The want of a knowledge of the studies opens up a real danger, of a one-sided training, and the want of a proper acquaintance with correct moral principles. As we have thrown over the old university system of preparation, we must replace those parts of it which relate to the training of moral qualities by preparatory studies in them, and supply the want of the training acquired by the study of Latin and Greek by at least their equivalent in other studies, in the requirements for admission. A low standard for the entrance examination will undoubtedly at first increase the number of applicants for admission, but numbers are not necessarily a criterion of success. They may, on the other hand, be a very decided sign of weakness. It is certainly true that that institution which offers the best advantages to the student will be the one which will in time be the most patronized, and that, in the long run, perseverance in demanding a high grade both for entrance and in the studies of the school will be crowned with success. But success does not always mean a large number of names on the roll-book. Numbers may, and frequently do, imply a low standard, the quality of instruction being lowered in order to get a large number of students. Such a policy must eventually result in disaster, both to the students and to the institution. It is referred to, only because, in the intercourse which I constantly have with members of boards of trustees of institutions all over the country, almost the only criterion of success which they seem to appreciate is numbers. I think I have never been asked by trustees of institutions, either in the east or the west, what the grade is; but I am constantly asked, as if it was the only criterion of success, what the numbers are. It is only when the

highest possible standard is maintained that numbers become a sign of strength. A high standard should be maintained and enforced both for entrance and during the years of the school. It is no charity to launch into the profession men barely prepared to enter it. This does not mean the exclusion of any slow-thinking students who would enter the profession for the love of it, but require a little more time to accomplish their work, but only of those who look upon it with little or with dilettante interest, are careless and indifferent about their work, or are unfit for it; and it throws the burden of labor on the preparatory schools, which must raise their standard accordingly if the entrance standard and requirements are higher. Not to maintain a high standard in the school is the greatest possible want of charity to the students, for a man insufficiently prepared to enter the profession is launched into it in a more or less crippled condition, and is liable to make failures which will be disastrous to himself and others. Almost the only way in which a high standard can be maintained in the school is by demanding a high attainment both to enter and remain in it. It would be better for the whole country if every school would adopt the principle of the old Paris School of Design, whose authorities were entirely satisfied if they turned out but one first-class man in a hundred every year, and who did not consider that their work was useless if they failed to produce one, if the cause of failure was the want of the attainment of that excellence at which they aimed to arrive. Our system differs from that of most other countries in the fact that the greater part of our schools receive no aid from government, and that comparatively few of them are sufficiently endowed to be entirely independent of the fees of the students, while in many cases they are mainly dependent upon this support. The temptation, in such a case as this, is to yield to a low standard, and it is sometimes done, but always with unfavorable results which are sometimes fatal. The engineers of this country owe a debt of gratitude to West Point, which for many years was almost the only engineering school in the United States. It has never lowered its standard, which as a government school it could easily maintain. They also owe a still greater debt to the Troy Polytechnic School, all honor be to it, which, being dependent on fees, has nevertheless maintained from its commencement one of the highest standards known in engineering educational institutions, and has turned out many a man who has been an honor to himself and his *alma mater*.

Of the methods of instruction, the lecture-method combines the

necessity for discriminating between what must be noted and what may be neglected—unflagging attention and rapid thought, with the manual power of noting it. The student ought to be taught to take his notes in such a way that they will not require rewriting. It must be said against this method, that it requires that every one should go at the same pace. With the printed text-book, or autographed notes, which answer the same purpose as lecture-notes, there is this great advantage, that the men can work fast or slow, according as they are capable of taking in the subject with more or less rapidity. They can go as fast as the lecturer, or, if they cannot follow, they have something to guide them. The textbook system is either expensive, or the book is behind the times; and it calls into play the eye and the memory only, while the lecture system requires the use of the ear, the eye, the judgment, and the hand. The autograph system of the French, which provides the student with a manuscript copy of the lecture-notes, to serve to some extent the purposes of the text-book, is an excellent one; for the student can read over the matter to be discussed before he comes into the lecture-room, and then expand his notes if desirable. Such publications can always be kept up to date, as they are not expensive, and they can be modified from year to year, while large editions of the text-book must be made in order to make its publication commercially profitable, and it necessarily remains for years unchanged. It may be said of all classes of text-books that they cultivate the memory almost exclusively, and that recitations made from them give the school-boy "feel," and do not cultivate the independence of thought and action which is necessary in the engineer. By the introduction of quizzes, or repetitions as they are called by the French, into any kind of instruction, the lame ones are brought up without the necessity of keeping back those who are able to work rapidly.

Every effort should be made to break up a system which uses the memory without the judgment; and, for this purpose, examination-questions should be those which involve the use of reason rather than memory, and, consequently, not on the details of subjects directly treated upon in the lecture-room, but only on the principles that they involve. I think too great stress cannot be laid upon this principle. It not only involves the habit of reasoning, but the habit of correct expression. A parrot can master words, but only a well-trained mind can master principles. I can remember very well, in my university experience, mastering page after page of subjects by

rote, because I was compelled to, without seeing until long years afterward the principles which those words involved. If both the instruction and the examination had been upon principles, the principles would have remained while the words were forgotten, and there would have been a large amount of intellectual force saved. It should never be forgotten that the object of education is not to go through a series of mental gymnastics which tend rather to overtax than to invigorate the brain, but to train the mind not only to receive facts and principles, but to be able to discriminate between what is true and false in what is presented to it for discussion.

One object of the instructor should be, not so much to answer the question of the student, as so to direct his inquiries that he will answer it for himself. There are, however, many cases where the allowing of the student to work longer would be a waste of time, and the instructor must therefore ascertain when the student is to be told, when he is simply to be directed, and when he must be left to himself to answer for himself. Just where the point comes when the instructor must answer a question instead of directing the student must be left to each instructor, who must know not only the man but the difficulties which each student is likely to encounter in his study of the subject.

There are a large number of principles which must be well taught, and insisted on at the very start. Accuracy of statement and the power of concentrating the mind for considerable periods on definite subjects are essential, and these are given by the study of mathematics. The constant care which is necessary in making demonstrations and proofs, brings about very largely an accurate habit of mind, but, on the other hand, has the tendency to make the mind demand demonstration, and to deaden or destroy the imagination. But a well-regulated imagination is indispensable to every engineer. It is necessary for him in his constructions, and in his application of old or in his development of new principles, and is a bad quality only when it is uninstructed and unregulated. There are many ways in which the accuracy of statement gained by mathematics may be effectually preserved without destroying the qualities of observation, of judgment and of imagination which are so necessary to the success of every department of engineering. These qualities are gained mostly in laboratory work and museum study.

The being compelled to seize at sight slight differences of shades of color, in making chemical researches; to make the distinctions in the tints formed by superposition of coats on coals with the blow-

pipe; to distinguish, like the spectroscope, mixtures of colors from simple colors; the observation with instruments of what the unaided eye cannot see; the reading of scales and micrometers, and correcting of very slight errors in mathematical researches, in which great accuracy is required, gives that intellectual power which first deliberates, then reasons, then generalizes, draws certain and fixed conclusions, and then makes their applications and permits at the same time of the legitimate use of the imagination. It is to such accurate laboratory work as this that we must look to prevent the deadening of the imagination which a too great predominance of mathematical studies would result in. These studies should, therefore, be supplemented by those in which there is something to do which requires the use of the imagination, and of the eye and of the hand at the same time. Another, and sometimes an equally desirable method, is to dwell at some length upon the lives of great engineers, the steps and methods by which they attained their success and celebrity. This will both fire the imagination and the ambition of the student, and lead him to think what he must do to arrive at similar success.

In order to see, one must be trained to look. To be able to see so accurately that the impression on the retina of the eye will be so perfect that it can be accurately transferred by the hand to paper, produces the power of self-reliance, promotes the habit of close thought, and searching for the principles on which everything that is to be represented in the record of instrumental observation is made, or in the drawing of the thing seen is to be constructed. It promotes accuracy of expression, both of the hand, the eye, and the tongue, and thus indirectly affects the moral well-being of the man by promoting a real love of truth. The powers of observation and of judgment are cultivated most thoroughly by watching the various colors in the blowpipe-work, the effect of their superposition, the way in which the coats come one after the other; by noting the characters of minerals to distinguish, not only their color, but the various tints of color, the luster, the shape, the usual form and the associations; by looking for the slight variations in crystals which distinguish one system of crystallization from another; by watching the differences of action in different motions in machines; by sketching a machine or its parts from memory; by ascertaining the exact time when the tool must be withdrawn from the lathe in order to make it possible to finish the piece without making it too large or too small; by the careful inspection of gauges and reading of scales, and the fitting of all detached parts within very restricted

limits—and in a thousand other ways, in which the pupil can easily be trained, and in which he unconsciously trains himself when he has been properly directed. To further aid the development of these faculties, he should be taught not only the careful observation of chemical, physical, and mechanical phenomena, but also accurate methods of recording them. This is greatly aided if the record is made on a series of carefully prepared blanks on which the phenomena are required to be described in a given order, and conclusions are required to be drawn from the data observed, which accustoms the pupils to systematic methods not only of observation but also of record. The peculiar advantage of eye-education is, that the eye sees and telegraphs the mind, the mind calls the judgment into play, which telegraphs the hand what to do, and tells the voice what to say. This education of the eye to observe the minutest differences in things, even if they have no bearing on the profession, gives the power of quickly detecting those very slight changes of action or of appearance upon the observance of which life may sometimes hang, and at the same time the discrimination which directs the remedy. Eye-training stimulates the brain to activity, which in turn makes the hand as well as the eye accurate. It is a slow process, acquired gradually and almost imperceptibly, and finally brings about that discipline of mind and hand which makes the hand work instantaneously with the eye, the eye directing while the hand executes the thought, the action being at the same time instantaneous and accurate if the education has been properly done. It also affects the whole moral character, for the indirect teaching of constantly making careful records of observations must lead not only to accuracy of method but to a real love of the truth for its own sake. Those who are so trained must be truthful as well as accurate, for accuracy means truth. The necessity for professional truth will thus become a part of the man and will influence his whole life.

Every savage diligently cultivates his powers of observation. His life depends upon it. The bent twig or compressed grass, the abrasion of the bark, the turning of a stone, and many other indications that would pass unnoticed except by one whose powers of observation were cultivated to the highest degree, tell him a clearer story than our printed page. It is only civilized man, who is so constantly served that he need not always depend upon himself, who must be urged to observe and made to see that careful training in that direction is necessary. Yet men always admire the application of these faculties when they have been cultivated. How many lives would

have been saved if, in cases with which every one of us is familiar, the powers of observation had been so cultivated that every slight change in color, which in iron or steel precedes its rupture, or the tremulous motion or slight crack, or many other small phenomena which give warning of danger, had been perceived beforehand, instead of allowing the accident to take place. Life is full of illustrations of the infinite harm done by careless observation or the entire want of it. To the engineer it is one of the most important faculties he can train, and the keen quick perception which he acquires in this way makes him foresee and correct the evil, stimulates him to prevent rather than cure, and gives him the skill which the world both appreciates and admires.

I do not think that the importance of drawing as a mental discipline is understood by most of our educators, nor sufficient time devoted to teaching it. It is generally treated more as a means to an end, a tool that one is to use as occasion requires, rather than as a method of acquiring mental discipline. I fear from the purely mechanical way in which it is taught that it is regarded more as a hand labor, than as one of the best means by which accuracy of thought and expression is to be attained. To be able to put into one of the universal languages, drawing, the idea that one has in his mind, so accurately that it is incapable of misconstruction, and that the thing or machine can actually be made from it, shows mental discipline and control of mental force already acquired. To see in space not only the thing itself, but the sections of any of its parts in any plane, and then construct what is seen, means great concentration of mental power, as well as confidence in the ability to use it.

I wish that I could make every child learn to draw just as soon as it is capable of holding a pencil. It is just as absurd to say that it requires a special talent to draw and design as it would be to say that there must be a special talent to learn the multiplication table. In a series of experiments on this subject, made some years ago, it was found that facility in drawing greatly aided the ability to master other things. The mass of the world looks but does not see, which is well illustrated by the fact that two persons of ordinary education looking at exactly the same object would describe it differently. The habit of sketching freehand and of making drawings tends to develop accuracy of the eye and its comprehension at the same time. To be able to see what one looks at is one of the most valuable attainments which it is possible for any man to have,

and there is no better way of gaining it than by carefully observing and then sketching what one has seen. The hand shows what interpretation the eye makes of what it has observed and how much of the principle has been mastered in the attempt to see. After a few unsuccessful trials the tyro learns just how far he can trust his eye, and then educates it up to a higher standard of accuracy. The eye must be aided by the memory, and thus the eye, and the memory, and the hand are stimulated and exercised at the same time. One of the best methods of cultivating this habit of accuracy is by making drawings from real objects and plans and sections in various directions, which not only gives the ability to be able to see what is looked at, but to look into and discuss and criticize in the section what cannot be seen from the outside. It also makes it possible to represent to others the most complicated thought, so unmistakably that the workman who never thought of the idea can execute it as well as the originator of it. The habit of accuracy once learned in mathematical demonstration and in the execution of sectional drawings results in a habit of mind which tells not only on the man but on all those who come under his influence. The constant dwelling on problems and things with the object of arriving not only at the truth, but at an absolutely accurate representation of the thing which is to be either executed or described, forms the habit of judging of things according to their real value, discourages superficiality in reasoning and shams in construction, and does away with the possibility of making the cheap substitutions and subterfuges which often pass for original and accurate thought and consequently develops the love of truth for truth's sake.

To gain facility of seeing what the meaning of a drawing is and to learn how to see what is to be shown in the sketch, models must be used by the beginner. The facility for drawing is greatly aided if after the sketch has been made the model is taken away and the same drawing is made from memory, with the same dimensions and proportions as nearly as possible. It is astonishing how quickly the eye is educated under such circumstances to take in not only proportions but minute detail. Freehand sketching taught in this way gives a power of memory which makes quick and accurate observation possible, and eventually leads to the habit of making the preliminary sketches in such a way that long after the object sketched has been forgotten the scale drawing may be made from the sketch with as much accuracy as if it had been made at once. Freehand drawing makes the brain express through the fingers what it could not express

by the mouth or by any amount of words. It leaves the thinker, the designer and the engineer to make his rough sketch, which can afterwards be transformed by the draughtsman, in any part of the world, whatever its spoken language may be, into the scale drawing, which is understood in the shops where the sketch might not be, and probably would not be understood, and thus saves valuable time of the engineer to be devoted to other things.

There is no doubt that in most of our schools sufficient importance is not attached to engineering design. I do not mean the making of pictures which look well and the copying of drawings or the sketching of machines or the putting together of different parts of different drawings, but the original designing of something for a specified use, and the making of such drawings as would be accepted in any shop for construction. This is perfectly capable of being taught, and that it does not require such a very high standard of intellectual culture to design things that can be executed is shown by the experiment made in one of our charities in New York, some years ago, when children who did not even know how to hold a pencil, mere street Arabs, collected indiscriminately as they passed along our highways, were brought together in schools in different parts of the city, and without knowing that it was difficult, taught first to draw and then to design, so that in a few months these children were not only making such drawings as could be executed, but were themselves designing patterns, and after a certain time learning to be capable to earn their own livings, originating the designs which they were executing in wood, metal or clay, thus firing their own ambition and raising themselves more rapidly than could have been done by any other means. The same experiment tried on older persons did not succeed so well, though there was a fair measure of success with them; but just one hundred per cent. of the children were successful in making designs which were not deficient either in originality or beauty, and which could be executed in the material for which the designs were made. The success with these children was mainly owing to the fact that they had no preconceived idea, either right or wrong, nothing to unlearn; that they did not know that what they were being taught was difficult; and that their ambition was fired by seeing that they could translate both things and ideas to paper, and that when they were told to create, their imagination was allowed to have the greatest freedom, though directed by the experience of those who taught them. If such results can be attained with the illiterate, what

might not be accomplished with those who have been carefully trained? A cultivated imagination, coupled with an educated hand, leads necessarily to design, and if the proper principles have been mastered, to that kind of design which leads to correct construction. It is a mistake, though a very prevalent idea, that designing requires some peculiar talent. The child with good eyes and hands and fingers that were capable of movement, who could not design and did not attempt at some time in his life to draw has not been born. As a usual thing, the first attempts at design are laughed out of the child by injudicious relatives. Who knows, if the rude drawing of the man or horse had not been laughed at and the child discouraged, but his first efforts had been encouraged and his hand properly trained, but that he might have eventually benefited mankind with some construction that would have caused a real advance in civilization? The imagination of the child is vivid but untrained, and can easily be directed into the proper channels. Imagination is only useless when uneducated and falsely directed. It is really the first step on the threshold of the unseen, and when it is well-directed it makes this threshold the open doorway to invention and discovery. Without the power of logical reasoning it is like the engine without the fly-wheel, which either wastes its power or expends a large part of it in tearing itself to pieces without producing any adequate return for the force applied.

Different degrees of talent are given to different persons, but fair success in design is certain to every one who is properly taught, while inspiration is given to but a few. Real progress is made by untiring effort. Advancement would stop if it had to trust only to inspiration. Well-directed effort, the result of careful training and study, is worth more to the world than the inspiration of the few, the very few geniuses who cannot be depended upon. A profession must have constant action, not spasmodic effort. To this end design should be taught in our schools, and illustrations of what may or may not be done with given materials, aiming only at what can be executed, should always be given. This instruction should be supplemented and illustrated by the best examples which combine beauty with strength, and which teach at the same time the laws of proportion. Ugliness is repulsive both to the eye and, in almost all cases, to the real utility of the structure. It is never true that the greatest utility and strength must be combined with want of proportion; proportion is another name for beauty, and it is true that the lines of symmetry and beauty are usually those of greatest strength.

There is no reason in itself why structures should not be beautiful instead of ugly, and with a proper cultivation of the imagination strength can just as well be combined with beauty as with ugliness.

The problems which are given in teaching engineering design should be selected with a view to excite the imagination as well as to develop the intellect; for there is no place where imagination is more required than in engineering design. Its cultivation not only brings about the possibility of designing beauty of form, but, when well directed, gives expedients and resources not only for current cases of work, but also in time of danger, for the education of the eye, together with the imagination, causes the powers of observation to be cultivated and sharpened, induces reflection and promotes action. The certainty that one sees what one looks at, and that he can with security not only interpret what he sees for his own immediate action, but that he has the power of transferring this observation or that thought into such a language that it can be comprehended by almost any one, promotes self-reliance in the highest degree, and at the same time makes independent thought, if, as a student, the engineer has learned to have a personal judgment. We may say that quick, accurate thought, and the power of expressing it with pen or pencil, is the key to the preservation of life and the invention of most of the improved appliances.

Making designs which are to be executed, produces a kind of mental training which is of the very greatest use in after life. Every line then made involves the thought, not only of how it is to be executed, but how it will look after the piece is finished. It implies concentration of thought as well as discipline of the mind, the hand, and the eye, which makes the accurate man. The possibility of seeing in space how sections of a given machine or structure will look, shows in itself a mental discipline and a power of concentration of mind that is not readily attained in any other way. To see the machine work before it is built, and to put the idea upon paper exactly as it is to be constructed, involves a mental process which is not only valuable to the man performing it, but to the community at large. It is the possession of this power, however it may have been attained, that has made the great engineers.

No one thing is more striking than the interest which students take in all kinds of laboratory work, whether it be chemical, physical, or mechanical, and the evident relief and recreation which it brings to them, as a change from purely mental labor to that which occupies the eye and the hand. For the time being, they become investiga-

tors, acquiring knowledge, making researches, drawing conclusions, watching reactions, and noting the working of different principles. Their work, to them, has just as much importance as if they were original investigators instead of learning how to investigate. Laboratory work—in which the closest attention must be paid to get the results, and the greatest accuracy is required to interpret them after they are obtained—not only gives to the student a capability for observation but a power of concentration of mind not easily obtainable in any other way; because the men are interested in obtaining results which they are themselves to interpret. They also learn the power of discriminating between things which are apparently alike and those which really are so, and of rejecting those which are of no special importance. There is, however, a real danger in this kind of work. We may cultivate such habits of precision in connection with those of observation by carrying them so far beyond their legitimate limits that a mental habit is formed which can see the point of a pin but is not able to distinguish the side of a house. We have seen such men in practical work, who, in their endeavor to avoid causes of error, introduced more sources of it than if they had made no attempt to avoid it. This, however, is a perversion of the true principle. The application of the facts learned in the class-room necessarily teaches the pupil not only to reason but to discriminate, not only between things which are true and false, but to differentiate between those which are of more or less value. It is not always easy to place the true balance between laboratory and class-room work, but it is just as necessary to train the eye, the ear, and the hand, as it is to acquire information which gives the brain the power of discrimination.

It may be said that in the schools one learns the theory, and one learns the practice in after-life—that the theory can never be learned except in the school, because in practical life there is no time for theory and it will all be practice. The old idea was, that any time devoted to laboratory work, more than a certain minimum, was wasted. The results of modern experience show that theory can be better taught in the laboratory than in the class-room, because its application follows at once on its announcement. It is thus fixed in the memory and ceases to be an abstraction. Besides, if the habit of applying knowledge is not learned in the schools it will hardly be learned outside of them, except by a long and painful experience. Both as mental recreation, and to train the powers of observation necessary for practical life, a certain number of hours each day should be devoted to laboratory practice of some kind, so

that the application of the theory may be learned at the same time as the theory itself. It would be useless to waste the time of the student in the machine-shop in teaching him to file or turn with great accuracy, or to make experiments and observations which can only be performed with sufficient nicety after long practice, because the principles upon which such work is done can be learned in a very short time. The execution of the work, however, can only be learned in the schools by such an expense of time as would not be justifiable; and while the principle will have its many applications, the engineer himself would rarely or never be called upon to perform the work. My own experience has been, that I derived as much benefit from the six months' training in the machine-shop, which was my recreation during the last year of my college course, as from my college education itself; but it is doubtful if the machine-shop would have been of such great use had it not been for the previous intellectual training. It is a mistake, into which many of our educators have fallen, to suppose that the time given to the training of the hand is lost so far as intellectual advancement is concerned. The two go hand in hand. No true education of the hand, nor successful laboratory work, can be done without training the mental faculties as well; and amongst the mental qualities acquired by such work are observation, which makes the accurate delineation of all the parts possible, and concentration of the mind upon the object being examined or discussed. In all laboratory work the principle should be to teach men how to do good work, but not to waste time after they have learned how. It is not the object of the professional school to make skilled workmen of the students. The moment they have learned how a piece of work ought to be done, they are capable of teaching those who occupy an inferior position how to execute it properly; and if this instruction has been coupled with the teaching of how to accomplish what is to be learned with the least amount of time and expense of material, they can teach other men to be skilful without necessarily being skilful themselves. It must be remembered that theory is usually acquired in the school, while practice is the question of a lifetime. It must always be kept well in view, that the object of introducing laboratory work into the schools is *in*-struction and not *con*-struction, and that when the latter is pushed beyond its legitimate limits the school becomes a shop, and that then a large amount of valuable time is sacrificed to the attainment of a skill for which there will be no use in practical life. What is wanted of the

engineer is not to do work accurately with his own hands, but to know how to do good work and to recognize it when it is well done, and how to correct what has been poorly executed.

Nothing is so powerful as the force of habit. Accuracy of statement, precision of ideas, probing things to ascertain what are the fundamental principles involved, are habits which can all be taught and encouraged in the school, and hence the formation of such habits in students should in every way be stimulated, and every possible impediment put in the way of shiftlessness, both in statement and in work. There is no better way, than by laboratory work, to form and stimulate such habits, and to make shiftlessness impossible. No better plan for doing this can be adopted than the giving of problems which the student knows to have been carefully worked out beforehand, and the results of which have been obtained from actual trial, or to execute drawings for which there is required not only accuracy, but the condition that they are to be so made that they can be used in the shops. I once saw some elaborate tests of iron and steel made in the greatest detail by officials who should have known better, where the measurement of the elongation of a piece of iron broken in the testing-machine was made by putting the two pieces end for end, and jamming them together in a vise, and then reading the elongation to thousandths. The school-training should always be such as to make such methods intolerable to the observer, however much they may be tolerated by the public. One of the greatest advantages of laboratory education is that it permits each man to be treated as an individual. No one who acquires knowledge or practice rapidly need be kept back if he works rapidly or pressed forward if he is mentally slow; every man works according to his own individual capacity and skill.

I have never been an advocate of doing in the schools by the ounce what is done in practice by the ton. Playing at work and teaching what must be unlearned is a vicious principle. As well teach carpentry with miniature chisels and hammers, or machine-work with a watch-lathe, as to teach engineering with miniature appliances. The advocates of this system claim that it gives the men confidence in themselves, but what is necessary is that other men should have confidence in them, which they certainly will not have when play methods are attempted to be put into practical execution. The schools should not teach anything which the man must unlearn, and this principle is being put into practice in many of our schools now. In the old methods great discredit was brought on the schools by this necessity

of unlearning. Better not learn at all, than learn what is to be acquired wrong, which is what the necessity for unlearning means. It is certain that practical experience will have to be acquired after graduation. The engineer has all his life to acquire it in. With real tools he can make a first step in the schools, so as not to be entirely ignorant when he commences practice, and he will then have a decided advantage over those who have not had much experience; but to take up valuable time in the schools, which is the only time in his life when he can learn theory, to acquire great skill by the practice of what he must afterwards unlearn or at least greatly modify is a fatal mistake. It would be better to devote the whole time to theory and leave all the practice to be learned under that usually hard taskmaster, experience, which will be much less hard, however, if there is nothing to unlearn.

We have found the greatest possible advantage from summerschools where the students actually work in the field or the mine and apply in practice the principles they have learned. This, however, is only another phase of laboratory work, and should be carried out with discretion. The object in most of such schools is to learn how to do, more with the object of directing others rather than of practical execution with one's own hands. There is the same danger here as in laboratory work, that time may be devoted to gaining excellency of performance rather than knowledge of the thing. I knew an engineer once who devoted several months to learning how to use the pick and shovel and worked 12-hour shifts with the men. He did acquire it and gained great physical strength, which was, perhaps, some compensation to him, but no such waste of time was justified by the engineering experience he gained. When the best methods have been exemplified attention should be turned to other things. Both time and energy are often wasted in these summer schools, which should be only the entrance gate into the practical school of real life. There is a real danger that in seeking to furnish what is practical and pushing the training of the shop or laboratory or field work beyond its legitimate limits, we may lose sight of that which gives the discipline on which success in life depends; that in teaching the how we may forget the why.

The museum, to those not well acquainted with its uses and with the derivation of the word, is associated with amusement, but the δμουσοι were the uncultivated and the unlettered, who went to the museums, the places devoted to literature and the fine arts, with

more or less of a hope of instruction as well as amusement. The museum itself was dedicated to the muses and maintained by the lovers of art, and to such a place, well arranged by the thoughtful teacher, the earnest student should be encouraged to go for aid in the study of his science, for recreation when overburdened, and for as sure a way of reaching his end as by oral instruction or book-learning, while the mere curiosity-seeker will be sure to carry away with him from the museum, if no other idea, that of order and classification. The museum is a necessary adjunct to the technical school, because it helps to familiarize the eye with colors, forms, associations, and proportions of things with which the student must in any case become familiar. Passing by and seeing, even without study, a mineral, a chemical, a part of a machine, or its whole, will familiarize him with color, shape, and proportions in a short time without effort, as much as many hours of study and more than the memorizing of a complete description from a book. Nor is it enough that the student should see only. He should be able to handle, to look over and to examine samples and specimens, placing them in every position, so as to become familiar with all their conditions, their weights, their dimensions, their proportions. No amount of book-knowledge would make it possible to distinguish minerals which are very much alike, or even if it did, could only be acquired at the expense of an unwarranted amount of time, not in any way proportioned to the value of the knowledge acquired. Museums in a technical school should be arranged on a principle which will bring out the relations of the things they are intended to illustrate. I am strongly an advocate of making all such museums, where it is possible, passage ways through which students must walk to and fro in going to and coming from their various class rooms, and of placing on the walls of the class and lecture rooms collections where the eye may rest, even though it be for only an instant in the intervals during the interruptions of thought. The eye in this way becomes unconsciously familiarized with objects which, from their previous familiarity with them, when they commence to study them closely, do not strike the terror into the heart which an absolutely new and unfamiliar thing sometimes does, thus making the subject less difficult. The order, the arrangement and the classification of the objects exposed carry with them a most useful lesson, which is unconsciously absorbed and teaches systematic methods better than almost any other way in which they could be taught. The museum thus becomes a powerful object-teacher. Its instruction is all the more emphatic because silent, and

the principles or facts which are demonstrated in the arrangement, all the more impressive because they are unconsciously learned. The student is thus without effort made to think that this or that must of course be true—we have always known it. It thus makes the acquirement of certain kinds of principles easy, where they might have otherwise been difficult if this powerful object-teacher, the museum, had not been brought into play. Familiarizing the eye of the student with good types makes knowledge of what is proportion almost intuitive. Beauty then becomes an element in his future design, to which his analytical knowledge will never allow him to sacrifice strength or utility. If the eye has been properly trained the drawing of a section stands out to him in relief, and he can distinguish by means of it what is behind or on one side as well as what is shown in the section or side of the relief which the drawing exhibits. There is always the danger in teaching from models, even though made to a scale, unless the idea of relative dimensions has been taught, that the real thing in its own proper scale will appear heavy and clumsy, and thus generate the habit of making things too light. This can always be guarded against by having the sections of parts of full size where they can be seen and handled. In my opinion too little is made of museum education in this country and too little floor space given to museums. They should be open throughout the entire day and be contained in well-lighted and well-ventilated rooms. The students should have free access to them at all times, and should be encouraged to pass as much time in them as is consistent with duties in other directions.

The idea of the students in old times was that the faculty was to be got ahead of by every possible means, and that each and every member of it was his particular and natural enemy. Fortunately for the success of our technical schools, this idea never was firmly implanted in them and is fast disappearing. Only the shadow of it remains. The true relation of the pupil and instructor is that of searchers after truth, in the pathway of which the teacher has advanced a little beyond the pupil and is helping him to follow. With such relations there will only be a feeling of friendship or even of intimacy, and never of antagonism. The professor becomes the pupil's helper to pass the difficult places, and his friend. To my mind, the system of elective studies, which is so large a feature of college education, has no place in the technical school. A most careful trial of it was made some years ago, under the impression that what would work in the college would work in the technical school;

but it was put down by the students themselves, who declined to take "optionals," since what they wanted was as full a knowledge of the professional subjects as possible, and asserted that if a study was optional that of itself was a declaration that it was not necessary, and that if two were "electives"* they needed both.

There are some things which are due from the instructor towards the student, as well as the student towards the instructor; and one thing is that, whatever else is done, the subject, however difficult, should not be presented in such a way as to make the student think it is so. In reviewing an old, as well as in presenting a new subject, the knowledge of the fact that the teacher is himself interested and thoroughly convinced, not only of its importance, but of the ease with which it may be acquired, goes very far towards solving many of the difficulties which the student may have. One thing is certain, that enthusiasm in the teacher begets it in the scholar. There is a magnetic power in the words of a really good lecturer that makes the attainment of knowledge seem really desirable. I remember very well, in my student days, being present at a series of lectures in a foreign language of which I could not understand more than half of the words, being so completely carried away with the enthusiasm of the lecturer that I was not only convinced that the subject, a mathematical one, was not so difficult as I had imagined, but was incited to much greater efforts to master it than I would have been from a dull lecturer had I heard the subject in the English language.

It should never be possible in the schools under any conditions for the student to acquire a smattering only, of anything. No one thing should be so severely dealt with by the instructor as the tendency to superficiality, to empiricism or to charlatanism in his pupils. He must, therefore, himself set the example by carefully mastering the subject which he has in hand, and showing that he himself is not only interested in the problem which he is elucidating, but really desirous that they should master it, which will not only beget in the student accuracy if he is accurate, interest if he is interested, but a desire on his part to master, as far as is practicable, the details, instead of acquiring only what is just sufficient for the passage of an examination. Such an example in the teacher stimulates a

* By "electives" are understood the studies, of which a certain number must be pursued by the student, and by optional the studies, outside of the elective list, which may be chosen or declined at will.

real desire on the part of the pupil for intellectual as well as moral honesty, which will tend to make the man in the practice of his profession honest not only in act but in thought. The enthusiasm of the lecturer is very catching, and no one distinguishes more quickly the perfunctory spirit than the student.

I remember, many years ago, being witness to two of the most flagrant abuses of the power of memory, both of which occurred in the mathematical class-room. In one of these classes the teacher required that the problems of Euclid should be mastered *verbatim*, so that not only the demonstration should be made in the identical words, but also that the angles should have the letters exactly in the same order. A B C was right, but C B A was wrong. The other was, where the teacher was satisfied if just half the problem was learned, and allowed his pupil to stop in the middle of an angle, if that was just half of the letter-press of the demonstration Such methods are beyond and beneath criticism. These men seemed to think that brain-development was gained by loading the memory with words. In both cases, as was natural, the teacher lost not only the confidence but the respect of his class, and after thirty-five years I still occasionally hear them spoken of, but without respect. It is greatly to be feared that the exercise of the memory only, is often fostered by the use of text-books from which the words and not the principles they involve may be learned, and that some of the same spirit is still extant. The mere acquirement of words, which consists of memorizing, may be beneficial to style in writing, if good examples are selected, but it is most prejudicial to originality of thought, and ought to be discouraged. Every student ought to be able to tell what he knows in his own language, for fear that he may have mastered words only and not ideas.

One of the greatest difficulties in the education which consists of injecting into the brain instead of developing its powers by drawing out of it, is that so many things are forced upon the mind that it is incapable of receiving, that it cannot hold them, and hence the system of cram, which brings about one of the very worst phases of intellectual decadence, the inefficient and the superficial man. The day when it was the object of the student to cheat the professor has long passed, but there are a few dishonest practices which remain in our schools, partly as a heritage of the past and partly as an indication that the old Adam has not been entirely removed. It is perfectly possible for the teacher to show that unfaithful work brings

its own reward in uncertain knowledge and untruthful ideas; that leaving to-day's work to be done to-morrow, makes it almost impossible to do either day's work well, and fosters the habit of being content with work poorly done, or of trusting to luck, which makes men unmanly and dishonest to themselves. Want of a proper respect for the teacher often grows out of the habit of demanding excuses in petty things, too much watching and not enough trust in the true manliness of the pupil. It is the student who makes the engineer, and if the manliness of the pupil has not been previously sufficiently developed, so that he remains a boy, while receiving a man's education, he has commenced his professional training too young, and should be returned to the preparatory school.

Cribbing is one of the dishonest practices of our schools, and the stopping of it has, for a long time, been the subject of the most careful consideration of both teachers and pupils. In the endeavor to produce a high standard in some of our schools, the pupils themselves have invented and carried out a police arrangement amongst each other against it, as being a dishonest and unmanly practice to such an extent that within certain limits they can be trusted, though too strong temptation should not be placed in their way. If the student is made to understand that "cribbing," written or spoken, makes him a living lie, there would be less of it, but I fear most students think, until taught differently, that "the end justifies the means." Another, and quite as dishonest, although a perfectly useless practice, is that of guessing. Almost every instructor knows when a student is going to guess before he opens his mouth, and in some of our schools this practice has been so far discouraged that the pupil will in a frank and manly way own at once that he does not know rather than to be so unmanly as to guess. The teacher who allows his pupil to guess is not only allowing him to do an unmanly thing, but is encouraging in him the dishonest idea that he may trust to inspiration instead of legitimate work and may at times impose on his hearers a pretence of superior knowledge which they may not discover. But this habit if not discovered at the time, must be eventually, and to tolerate it is to encourage him not only in present deceit, but in cultivating for himself uncertain and unfaithful methods. Whatever else is done, deliberation must be encouraged, and haste discouraged. One thing must be thoroughly impressed upon the mind, that no good work can ever be done in a hurry. The effect of hurry in intellectual work is like that of patchwork in

a machine; it may be made to hold together, but it never can be symmetrical and there will always be the danger that at some time some of the parts not well matched will give way, and the result will be disaster.

If it is possible to check the desire, which has been fostered so many years by our systems of education, of acquiring knowledge simply for the purpose of passing an examination, and not in the interest of mental development, it should be done. The highest compliment ever paid to the course in the institution which I represent was that of a young man who was then already a distinguished mathematician, and who had intended to devote his life exclusively to the study of mathematics, who came to the school, as he said, not that he ever expected to apply what he could learn there, but for the mental discipline which, from careful examination of the course, he was sure he would gain from the arrangement of the studies. We ascertain what progress the men are making by examinations which are conducted in three ways; they are practical, written or oral. While there are advantages in the oral method of examination for well-prepared men who have complete control of themselves at all times, practical work or the written examination seems to me to be the fairest, provided that the subjects to be discussed require some originality of thought in the student. I have always been in favor of giving problems which involve the principles taught, but which are not directly treated in the course of instruction. While it is not always possible to do this, yet the principle can be carried out. A written examination, made in this way, is a fairer test of the capacity of the student than an oral one can possibly be. It forces him to classify his knowledge and prevents him from the parrot-like use of his memory. In laboratory examinations, where there is something to be done, this is always the principle on which the examination must be conducted, and it seems to me to be the best method for the theoretical part also.

It is altogether improper to make the passing of examinations the only test of a student's qualifications. I have in mind two men, one of whom I knew to be qualified, but who failed in his examination from nervousness, resulting from overwork; and one whom I knew not to be qualified, who passed with great credit. Under the old law the best man had to pass his examination over again, much to the mortification of himself and his professors. But the luck of the other one followed him for only a short time, and in his future career he

showed failures both of ability and character, while the man who failed from nervousness occupies and always will occupy an extremely creditable position in the profession. Yet the student under examination has the right to the credit for what he does, but there should be such checks and safeguards as will give to the examination a coefficient only.

A point which requires attention, and should always be a subject of instruction, is the method of estimating the values of different kinds of work as well as their stability. The item of cost is the first which confronts the engineer in life. The question of the possibility of construction constantly occurs in the experience of old engineers and is one of the first questions which presents itself to the young man entering his profession. If he has had no instruction upon such questions, he must find this out for himself, but it would be much simpler and better if instruction were given in it as well as in other business matters connected with the profession. A knowledge of general bookkeeping should be required for entrance, but as the details of keeping accounts for each specialty can only be applied to it, they should be discussed as the matter is gone over in the lecture, and their peculiarities presented.

It is not to be forgotten that one of the most important things for the engineer is to avoid not only waste of material but of time. One is as much a part of the cost as the other. One of the best methods of teaching this in the schools is the setting of tasks to be accomplished in a given time, and in the shops of making certain things with a given amount of material. This may be done to ascertain what the capacity for work of the individual is, and by stimulating the habits of attention and rapid and accurate thinking, to bring up those who are mentally slow to work in competition with those who are more brilliant and rapid. There is no doubt that the rapidity of thought may be stimulated in the slow individual without of necessity decreasing his accuracy. One of the greatest advantages of laboratory work of all kinds is that it is full of the possibilities of such problems. The difficult problem to deal with is with those who do not think, but such material as this would be soon weeded out of almost any of our technical schools.

The health side of the question must not be forgotten. No education, however complete, avails anything unless the body is healthy, and there should, therefore, not only be such an arrangement of studies that recreation should be constantly afforded by change of

work, but there should also be some provision made for moderate bodily exercise, which is necessary to every healthful mind. Whatever excess of energy the student has, its use should be encouraged out-of-doors. I have always believed that recreation for the student in school hours could always be accomplished perfectly by judicious change of work, and that exercise, to be healthful, should be in the open air more than in buildings. There is always danger that exercise in-doors will be carried to excess. If gymnasiums are furnished, or out-door sports encouraged, the exercise should always be carried on under the direction of a wise physician.

I think, as I have said on a former occasion, that the characteristic of our American schools is, that the instruction given in them leads men not only to form a judgment, but an independent one;* to form opinions for themselves, and yet not to pass by the results of the experience of others. This is exactly the difference between the teaching of our own schools and those of some foreign countries. There the judgment is rather that of the chief, with whom it is often dangerous to differ, than of the individual, and hence the young man knows but little of mental liberty and independence. He is expected to receive his opinions on authority, not to form them for himself, and hence there is a tendency to a want of self-reliance and a habit of hesitating about advancing an opinion which may be overruled, which is fatal to self-reliance. The value of any education, however, does not depend so much upon the facts learned, the quantity or quality of the opinions absorbed and the amount of knowledge gained, as on the power of reasoning acquired; not the amount actually stored in the mind, but the knowledge of how to use it and confidence in the ability to do so, and where to go to find what is not known, or get the information relating to any series of facts when it is wanted. Hence the student must be taught to think for himself, so that as an engineer he may have his own independent judgment and not be constantly asking himself where shall I go to get an opinion to follow.

The old theory that the beautiful statue exists in the stone, and requires only skilful sculptor to extract it, is certainly true of men. The materials are all there: it requires only the skilful educator to bring them out; and just as the sculptor sees the statue in the stone or the mechanic sees in the irregularly shaped piece of iron the finely-finished part of the machine, working in its proper place, so

* Discussion on Technical Education, p. 98.

should the educator be able to train young men in the duties of their profession, making them symmetrical both in their mental and physical acquirements, bringing out the faculties they have, and developing those that require expression or expansion, so as to make the symmetrical man. With men in large classes this can hardly be done, hence numbers may be a sign of weakness, and really are so, unless the standard is high and the force at command is sufficiently large to ensure the treatment of each man as an individual, and not as only one of a large number.

Unlike the ordinary university graduate, the engineer leaves the schools for practical life-work. He has to deal at once with problems that require the immediate use of nearly all the faculties which can be educated, and not a part of them. His hands are required to do, his ears to hear, and his tongue to direct; and without the proper training, the systematic equalization of the powers required to effect this, he becomes in his practical life a one-sided man, and is likely to remain so unless he has the unusual power of first seeing and then correcting these defects.

The success of the engineer in life depends as much on his character as his education. The old theory of only putting knowledge in, did not develop character to any very great extent, but the real purpose of bringing out what there was in a man, and cultivating any germ of talent, however small, not only does develop the character, but the mental faculties, which will always seek the practical application, as the old process never did or could. The administration of affairs, and the management and control of men, are as necessary to the engineer as the application of his technical knowledge. These can only be acquired to a very moderate degree in the schools. They are not the places where commercial rules are to be learned. Relative facts only can be taught there. The positive relations of labor, material, wear and tear, are likely to be the same everywhere when the conditions are the same, but commercial relations will be different. To be a good executive officer cannot be taught in the schools; it is to be learned by contact with men, whether it is to be the control of intellect or of mere labor; but the reasoning powers which are developed in the constant application of facts which are learned to practical work, and which must be made by the eye, the ear, and the hand, give the power of discrimination and reasoning which should make it more possible and easy for men scientifically educated to acquire this control than for other men. It is as much a knowledge of one's own as of other men's

character that leads to the ability which successfully manages men. Kindliness of heart and a real interest in his men is what they discover rapidly, and if this is coupled with firmness the questions between the men and the employer will rarely if ever cause real difficulty.

As the engineer has to do with natural forces, which control the phenomena he has to study in nature, and which he must, in a certain sense, control in his working, he must learn in his studies to discriminate between those forces which are essential to his work and those which are only a by-play in it. This careful observation leads to the neglecting of unimportant things, and to the placing prominently, not only in the mind but in the actual execution of the work, of those things which are the controlling ideas and forces by which the work is to be governed.

There is a great advantage in scientific education. It deals with facts as well as theories. A thing must be true, or it is false. In the continued search after facts, the accurate statement which truth requires not only develops the mind, but the character must be influenced more or less to the desire for high and still higher aims in life. No search after truth can be made without a theory. The results of research show this theory to be true or false, and, as the object is truth, the moment the facts will not bear out the theory it is discarded. How different the result of metaphysical discussion; everything there is bent on sustaining the theory; subtleness of reasoning becomes its aim. It is true there are no facts directly affecting human happiness in the results of its discussion, but the higher character must be formed by the continual search after truth occasioned by the constant desire to ascertain what is fact.

There are things in the profession of an engineer which make him necessarily more accurate than other men. The knowledge that, in all branches of the profession he may be called upon at any time to face danger, and is liable at any moment to be brought into contact with facts, a mistake in the detection of the minutest character of which may at any time cost his own or the life of another man forces the habit of coolness under trying circumstances and should make him truthful to himself and to others. The knowledge that he cannot run away, but that it is his duty to remain in times of trial, makes him brave and capable of facing danger as he would face any other fact in his professional experience. The highest devotion to duty leads to the result of the world's being better for one's having lived in it, and ambition should be towards contributing to the

sum of human knowledge rather than towards money success in professional practice, but the attainment of both is quite possible.

The great engineers who made the end of the last century and the beginning of this famous, spent the greater part of their lives in acquiring the knowledge which we teach every student in our schools. We do not say what these men might have been with a technical-school training, but certainly the young graduate of to-day starts with very great advantages over them. Whether or not this century, with technical schools, will do as much for engineering as the last century without, remains to be seen; but it is certain that the knowledge of these subjects is much more diffused than it formerly was, and that mankind as a whole is receiving a benefit which was not possible when there were so few engineers that every one who carried out any large undertaking became of necessity a great man.

There is a great advantage for the student in becoming acquainted before he enters his professional career with the words in which the mechanic expresses his thoughts, or, in other words, the terminology of the shops. It gives confidence on both sides, to the mechanic as well as the student, makes the language of each intelligible to the other, and makes the mechanic feel that he and the engineer are on a more equal footing. I have, on a former occasion,* entered a plea for the laborer, and especially for the skilled laborer, who has learned by long experience to correct by his eye and to do with his hands in the workshop what really entitles him to respect. The man who depreciates and looks down upon manual skill, underrates the first principles of engineering. It is the contempt of it which has kept the world back, and has been indirectly the ruin of many an intellect which, if its ideas had been allowed to come out of the brain through the hands, might have contributed materially to the advancement of the world. There is a great difference between this century and those which preceded it, in the respect into which skilled mechanics have risen in the eyes of the community, and the thorough appreciation of their work which the world to-day accords. It has usually been the case that the great inventions which have startled the world at the time, and have resulted in great progress amongst communities and nations, have come more from the training of the hand than they have from the old-fashioned training of the schools.

After graduation there is apt to come a period of discouragement, when lucrative employment does not come; and this is the trying time when character is developed. To find out how to apply what he

* Discussion on Technical Education, p. 100.

knows, to turn his theory to some use, to find out exactly what niche he is made to fill in the world's mental and industrial progress, is the most difficult and trying problem of a young man's life. The old motto, "I will find a way or *make* one," should be the principle of every young man starting in life. There are plenty of professional rewards for those who have the courage and the manliness to win them. Those who have had the greatest amount of influence in procuring places at the start have not always been those who have been the most successful in their professional career. The greatest men have been the engineers of their own fortunes as well as great engineers in their own profession. Discouragement never accomplishes anything, but continual trying does. The power of not knowing how to be discouraged is enviable in all men, and this is one of the few things of which it is well to be ignorant. It is very essential that the young engineer should not despise the day of small things, nor the careful observation of them. It has generally been by slow steps, and the careful observation of minute things, that the greatest discoveries have been made. Nothing is too small for the intelligent engineer to investigate; behind it may hide some great principle.

We have in this country multitudes of institutions whose only province appears to be to crowd the brain with theories and facts already known. How few are those whose function it is to add to the sum of human knowledge. We ought to train, if possible, our engineers not only to learn and apply facts already known, but to seek to add to the sum of human knowledge by investigations into what is not known. It is a great lack in our methods of education in that there are no schools for training investigators. Any one can get results, but few men can tell what they mean or what their relations are to each other, or to similar facts. To the education and training of men who have this faculty, which is a very uncommon one, time ought to be devoted, for the quality of mind which they have is very rare, and can only be raised to its greatest value when it has been carefully trained. It is astonishing that the "adding to the sum of human knowledge" has attracted so little attention, and that so few endowments have been made for the purpose. What more permanent monument to one's memory could be devised, or what better use of his fortune could be made, than to have it said of him, as was once said, I have made these investigations "a daily pensioner on the bounty of one who entered into rest more than five hundred years ago."*

* A Plain Commentary, vol. 1, p. xvi. Philadelphia, 1856.

The memory of such a man would never die, and truly his good deeds would live after him.

The introductions into our common schools of such methods as have made technical schools successful would, in my opinion, go further to solve the labor-troubles than any other means yet proposed. If a part of the time, spent in the schools in learning the "three Rs," had been spent in directing the pupil how to use his hands, I am certain that the result would be greatly to the benefit of mankind as well as the individual, and I am also sure that if part of the large amount of time spent in the preliminaries of learning to write, had been spent in learning to draw, the pupil would have learned not only to write better, but to express himself in a language that could be understood by everybody. Skilled labor is, of necessity, conservative, and not likely to lend itself to either lawlessness or riot. Unskilled labor is, in general, ready for revolution on the theory that almost any change must be for the better. By our present method, the children of the hewers of wood and the drawers of water are, to a great extent, unfitted for the situations in life which they occupy, and not fitted for any other. They cannot be blamed for the idea that they should occupy positions for which the school-training has not fitted them, for since the common school has taught them to despise all labor but that of the brain, and this function only has been trained, they think themselves fitted to occupy any position. They have not been taught to honor manual labor. They see in it only toil and small rewards. How different might it have been, had they been taught, as inspiration enjoins, "Whatsoever thy *hand* findeth to do, do it with thy might."* They have read this plain injunction many times, but their education has taught them to substitute *brain* for *hand*, and the precept is as unintelligible to them as if it was written in the original Hebrew.

The world is full of laborers. We are all such, but the uneducated classes are unable to obtain their living in any other way than by what is called manual labor. Against the reproach connected with this designation, I must here enter a protest. These men *toil* with their hands, because they do not know how to use them intelligently. It is frequently said that anybody can dig, which is true; but the production of the greatest economical value of the labor expended in raising a certain amount of earth on a shovel with the hands may be nearly double as between one man

* Ecclesiastes, ix, 10.

and another, if the useful effect of the kilogrammeters of force expended is to be considered. I hold that the appellation "manual labor," which usually carries with it a certain stigma, is only used because the laborer does not know how to use his hands. His labor is toil, the mere exercise of brute force, because it has not been educated, and he feels degraded by it. When it has been educated he no longer despises his labor; it has become the exercise of skill, and he both respects himself and demands and receives respect from others. Manual labor skilfully performed is often as worthy of regard as intellectual achievements, and we acknowledge this in the respect we have for and the wages we pay to the skilled mechanic. The world is lost in admiration at the works of its sculptors, painters, and musicians. Why should painting, sculpture, and music monopolize the rewards of manual skill? Why should not every human being be taught to use his hands and eyes? It is certainly more useful and more necessary than the memorizing of many of the things taught in our public schools, which do not give equal mental discipline, and cause the man and boy who have no reliance but the sweat of their brow to gain their livelihood, to look down on the labor of their hands. Why should hand-work, if well done, be any more disgraceful than the work of any other part of the body? How much are we to blame for the discontent of labor when we ourselves have indirectly taught in our public schools that the labor of the brain is the only labor that is not disgraceful? We have driven a great part of what would be our most intelligent labor from the fields to fill the precarious positions of clerks in our large towns and cities, when the educated farmer is what the country most needs, and the one upon whom the country most depends in time of real danger. There is little reproach to be cast on the methods of our engineering schools. Every one of them may be improved in detail, but their general method has always been successful, because it tends to the harmonious development of the whole man. Why should we not adopt a similar system in all of our schools?

The only true principle of education is the symmetrical development of all the powers which the intellect governs, or, in other words, the harmonious development of both the intellectual and physical faculties.

TROUGH-LIXIVIATION.

BY OTTOKAR HOFMANN, ALAMEDA, CAL.

(Boston Meeting, February, 1888.)

IN tank-lixiviation, the extraction of the silver from chloridized ore by solutions of hyposulphite salts is performed by filtration. The ore-particles are kept stationary, while the solvent moves down through the mass of ore. The quickness of extraction, other conditions alike, is in direct proportion to the rapidity of the movement of the solvent through the ore. The solution, if left in contact with the ore without moving, displays but very little solving energy. If the filtration is interrupted for ten or twelve hours, and thus solution and ore are left in complete contact for that length of time, it will be found that, when filtration is started again, the outflowing solution is but very little more saturated with silver than it was at the time of interruption, and that the ten or twelve hours were almost a perfect loss in the total time of extraction. Notwithstanding the long contact, the solution had not become saturated with silver chloride to its full solving capacity. A rapid movement of the solvent through the ore is essential to a quick extraction. This fact is well known; and the endeavor of leachers has been to hasten extraction by increasing the rate of filtration. Siphons, vacuum-pumps and other devices have been used with more or less success; but none of them have given full satisfaction.

I have found that, if chloridized ore is brought into rapid contact with a proper volume of moving sodium hyposulphite solution, the silver chloride contained in the ore dissolves almost instantly, and that it is rather the volume of the solvent than its concentrated state which produces this effect.

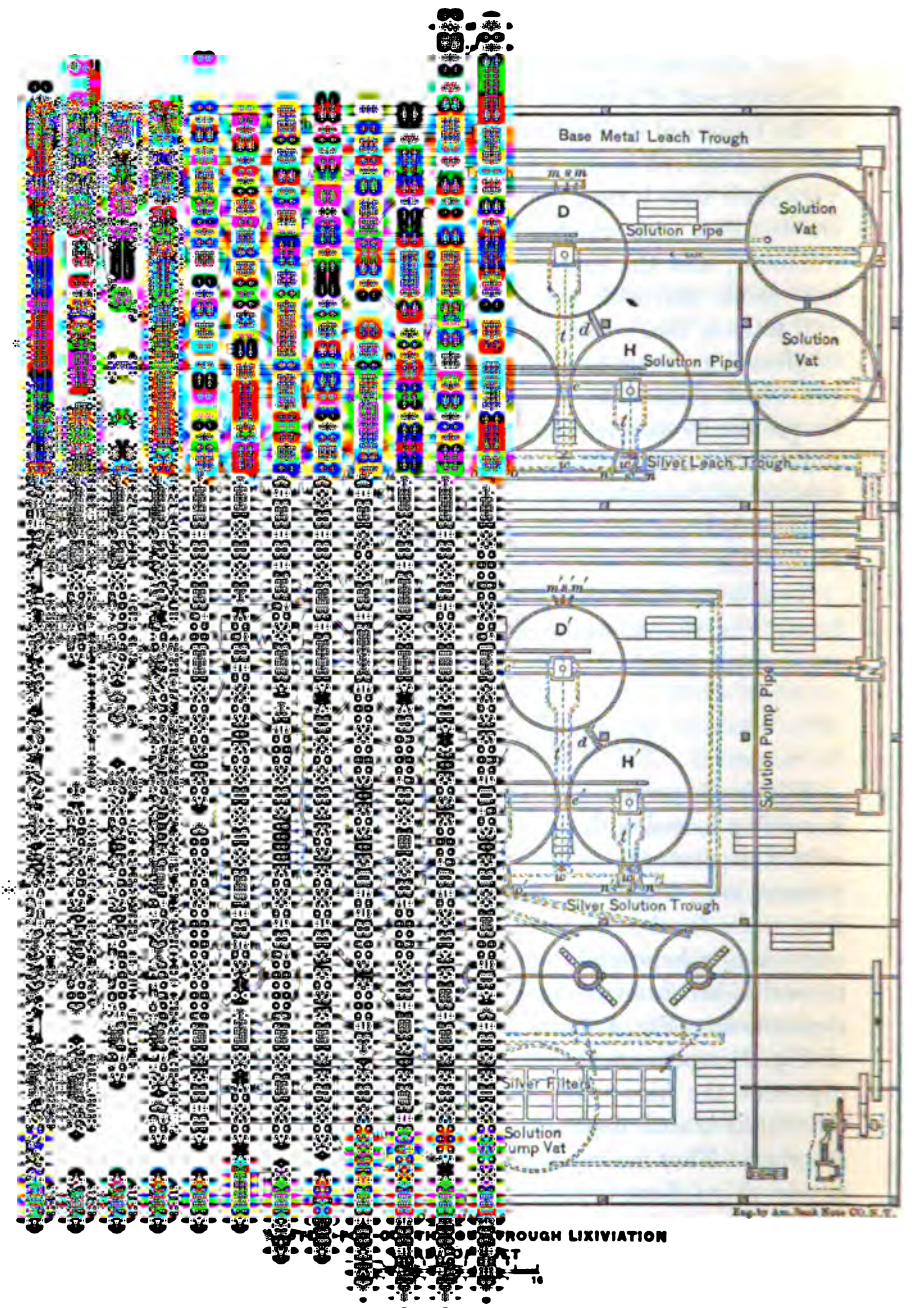
Such favorable conditions cannot be attained in tanks. The rapidity with which a certain volume of the solvent can be brought into contact with the ore-particles is limited by the speed with which the solution descends through the ore; and thus the leaching-time in tanks cannot be shortened beyond the limit set by the filtering-capacity of the ore. In a trough, however, these favorable conditions can be attained by gradually introducing the ore into the moving stream of the solvent. The ore can thus be brought into

rapid contact with any desired quantity of the solvent, and moves in and with the stream. The effect is astonishing. Ore charged at the upper end of a trough of moderate length, will leave the trough at the lower end desilverized, as tailings, while all the silver chloride will be dissolved in the solution. This is accomplished during the very short time it took the pulp to rush through the trough. The shortest time with which I have experimented was three-quarters of a minute; and the extraction was complete. Further experiments may prove that with certain ores the required time for extraction will be even less. But in order to obtain satisfactory results, it is of great importance to maintain a certain proportion of solvent and ore, which proportion depends on the nature of the ore and has to be previously ascertained. Stronger solutions of sodium hyposulphite do not always lessen the required volume of the solvent. In some of my experiments I even found that I had to use much less of a one-half per cent. than of a one per cent. solution to produce the same effect.

This is in short the principle upon which trough-lixiviation is based. In the following paragraphs I give a description of the new system itself.

Arrangement and Operations.

Solution is performed outside the tanks in troughs, while the ore is moving in and with the stream of solvent, and the tanks are used only to separate the solids from the liquid. The system is a continuous one; but as the lixiviation process requires two solvents, first, water for the removal of the base-metal chlorides, and then a solution of a hyposulphite salt for the extraction of the silver, it has to be divided into two departments: the base-metal and the silver department. Fig. 1 shows a complete arrangement, of which part of Fig. 21 represents the vertical section. The upper series of tanks represent the base-metal, the lower series the silver department. The tanks in each department are placed on the same level and close together. They are connected by pipes, *a*, *b*, *c*, *d*, *e*, *f*, *g*, *h*, in such a way as to form a perfect circuit. These connecting-pipes are placed a few inches below the rim of the tanks and also on a level. The diameter of these pipes depends on the daily capacity of the works, and the proportion of solvent and ore to be used. Each tank is provided with outlets, *i*, *k*, *l*, *m*, *n*, *o*, *p*, *q*, which are on a level with the connecting-pipes. They lead to the base-metal solution-



TROUGH LIXIVIATION

trough, which is placed below the bottom-level of the tanks. Into this trough leads also the outlet *s*, from under the filter-bottom. Each tank is also provided in the center of the bottom with a discharge-hole, through which the ore is sluiced into the silver-leach trough. The pulp (roasted ore and water) is conveyed from the furnace-house to the base-metal department in a triangular trough, which enters the building near the roof, descends in zig-zag, and branches off in two troughs which pass over the two rows of tanks. Over each tank the trough is intersected by a square box, the bottom of which contains one or more plug-holes, in order to permit the charging of any desired tank. By means of slide-gates the stream can be directed to flow in either of the two branches. The sluice-troughs *t*, beneath the bottom of the tanks, all discharge into the silver-leach trough, which leads to the silver department. Here, the tanks are constructed and arranged in exactly the same manner as those in the base-metal department. The solution storage-vats are placed in the base-metal building.

Base-Metal Leaching.

The pulp of water and ore, after passing a grinding-machine, in which the lumps are mashed, and of which I will speak further below, enters the triangular base-metal leach-trough, and is thus conveyed automatically to the base-metal tanks. The first tank to be charged is D; but before commencing operation, the following preparations have to be made. The slide-gate *x* is closed, to direct the stream into the first branch-trough, and the plug-hole of the square box above D is opened. The connecting-pipe *d*, between tanks D and H, is closed by a plug from the inside of D. The outlets of all the tanks are closed except the outlets *n* of tank H. Likewise all outlets from under the filters are closed. This done, the pulp is admitted to flow into tank D. When the solution reaches the level of the connecting-pipes, it will flow through *c* into tank C, and when C is filled, into tank B, and so on until the last tank H is filled, when the solution will leave the tank through outlet *n*, *n*. As soon as tank D is filled with ore, the pulp is made to flow into tank C. The connecting-pipe *c* is closed, and thus tank D is disconnected from the circuit. The outlet *s*, below the filter of tank D, is opened and the solution is allowed to drain into the base-metal solution-trough. When the solution disappears below the surface of the ore, water is admitted, to press out the solution ab-

sorbed by the ore. When this is done, sodium hyposulphite solution is applied to press out the water. As soon as the liquid flowing out through a filter-outlet *s*, shows indications of silver, the outlet *s* is closed, and the charge is ready to be sluiced for silver-leaching. While tank-charge D is under the described treatment, which does not take much time, tank C is gradually filling with ore. When filled, the pulp is made to enter tank B, and tank C is disconnected from the circuit, and its charge is treated in the same way as that of D. Charge D being sluiced out with solution, the filter-outlet *s* is turned from the base-metal trough into the silver-leach trough, the hose-clamp is opened, and the solution contained under the filter is allowed to flow out. This accomplished, outlet-hose *s* is closed and returned to the base-metal solution-trough. Then the plug of connecting-pipe *d* is removed and tank D is restored again to the circuit. D being empty, the flow in the base-metal solution-trough will cease until the tank is filled again with base-metal solution.

Separating Ore from Solution.

In an article which appeared in the *Engineering and Mining Journal* of Sept. 10th, 1887, I gave a short description of my new system of lixiviation, which brought forth some criticism. It was stated, that in order to obtain clear solutions, it would take a large number of tanks, more than double the number used in tank-lixiviation. This, however, is not the case. The physical condition of ore before and after roasting is different. When raw, a large portion of the material is in a very fine condition, which will float in water for a long time, and thus offer difficulties to separation; but when roasted with salt, the ore becomes sandy. The fine ore-particles congregate to coarser, porous grains, which behave like sand, and form but very little slimes in water. The separation takes place in a few hours. I have observed, that, while the pulp was pouring into a tank of fourteen feet in diameter, the solutions near the sides, and furthest off from the inflowing stream, became almost clear to the depth of two or three inches, which indicates a quick separation.

Across each tank, and at a right angle with the direction of the flow of the solution, there are hung six or eight curtains of coarse cloth, which reach near the bottom. They are kept in position by rocks fastened to the bottom-end, while the top-end is nailed to wooden rollers resting on the rim of the tank. The object of these

curtains is to prevent the formation of streams in the tank, thus quieting the liquid, and to assist separation by the large surface which they offer, and to which the fine particles will adhere. When preparing a tank for charging, the curtains are well shaken, and rolled up to prevent them from being packed in with the ore.

The surface of the solution in all the tanks will assume the same level, and the movement of the liquid through the connecting-pipes will be gentle, and will not have the tendency of stirring up the settled material. The curtains will break the gentle current near each connecting-pipe and prevent the formation of streams in the body of the liquid, so that the flow from one tank into the other will be caused by the surface being raised by the influx. Though theoretically not quite correct, we can practically consider the outflow to be caused by displacement. In other words, the solution which flows out from the last tank had to remain in the tanks for such, or nearly such, length of time as is required for filling all the tanks used for clearing the solution, and, therefore, a moderate number of tanks will be sufficient to obtain clear solutions.

As the chloridized ore "slimes" so very little, the quantity of slimes which will have accumulated in a tank at the time when its turn comes for charging, will be moderate. When the stream of pulp is turned in, the coarse material will go through the thin layer of slime, displace it to a great extent on the filter, and become partly mixed with it, so that the slimes will not interfere so much with filtration as might be expected. Of course, it will not be quite as free as if the ore had been charged in the usual way; but the use of the filter is so limited in trough-lixiviation, that a less rapid filtration is no obstacle, especially as in case of need a vacuum-pump can be used.

Solubility of Silver Chloride in Base-Metal Solution.

Silver chloride dissolves in solutions of metal chlorides, especially if concentrated and hot. Solutions of moderate concentration, however, do not dissolve any, at least, not in so short a time as is required in trough-lixiviation. I made the following laboratory test: Freshly prepared silver chloride was introduced into solutions of sodium chloride which had a temperature of 120° F., and was left in contact for five minutes, during which time the solution was vigorously agitated, and then filtered. The filtrate was tested for silver with sodium polysulphide. I found:

Solution containing	2	per cent.	NaCl	at	120°	F.,	dissolved	in	5	minutes,	no silver.
"	"	3	"	"	"	"	"				"
"	"	4	"	"	"	"	"				"
"	"	5	"	"	"	"	"				"
"	"	6	"	"	"	"	"				"
"	"	7	"	"	"	"	"				some silver.

The filtrate showed a faint coloring by sodium sulphide. When using an 8 per cent. solution the coloring was still very faint. Molten silver chloride poured into a 5 per cent. sodium chloride solution at 100° F., decrepitated into a fine powder; but the filtrate did not show any reaction for silver.

In trough-lixiviation we can regulate the grade of concentration of the resulting base-metal solution at will; and it would, therefore, be an easy matter to produce at once a solution which is sufficiently dilute not to dissolve silver chloride. But the effect is different if chloridized ore is dropped into water. Some silver will dissolve, even if the resulting solution is sufficiently dilute not to dissolve silver chloride as such. It seems that by introducing the ore into water, that part of the water which penetrates the ore-particles first becomes sufficiently saturated with metal chlorides to dissolve some silver chloride. This, however, cannot remain dissolved if the resulting solution is sufficiently dilute, and is precipitated again. Therefore, by producing at once a sufficiently dilute base-metal solution, we will have some silver chloride suspended in it. The rapid motion of the pulp in the trough and its intersections is equal to a vigorous shaking in a flask, and will greatly assist in collecting the finely-divided silver into flakes. These flakes will settle together with the slimes, and are extracted by the subsequent silver-leaching. Should it be found in some cases, that the out-flowing base-metal solution still contains some finely-divided silver, it can be conveyed to a large pit outside the mill, where the small balance can settle. It is evident that trough-lixiviation offers the means of regaining the main part of the silver dissolved by the wash-water, without subjecting the latter to a separate treatment.

Should it be desired to produce more concentrated solutions (which may be the case if the roasted ore contains enough copper chloride to make its saving an object), nothing prevents the operator from producing such solutions and proceeding afterwards in the usual way.

If concentrated base-metal solutions are produced, not all of the base-metal chlorides will be dissolved, and an additional slight washing of the tank-charge will be necessary. My experiments in

this direction are not complete enough to form a positive opinion; but it seems that the perfect and quick dissolving of the base-metal chlorides in water is subject to the same principle as the dissolving of silver chloride in a solution of hyposulphite salt, viz.: that a certain volume of the solvent is required to produce the effect. I made some tests in the laboratory, one with ore from the Cusihuiriachic mine, and one with ore from the San Francisco del Oro mine of Santa Barbara, Chihuahua, Mexico. The ore from the latter mine contained over 23 per cent. zinc-blende, besides some galena and iron pyrites. Both ores were roasted with 8 per cent. of salt. Twenty grammes of each were placed on a paper filter and leached with water. When all the soluble base-metal chlorides had been extracted, the filtrate was weighed, and it showed that the amount of water required for the Cusihuiriachic ore was three times the weight of the ore, while for the San Francisco del Oro it was seventeen times the weight of the ore. By treating the same quantity of ore one and a half minutes on the trough principle, no perfect extraction of the base-metal could be obtained, until the same proportion of water and ore was used as that required in common leaching, viz.: 3 to 1 for Cusihuiriachic and 17 to 1 for San Francisco del Oro. Both these ores are very base. In Cusihuiriachic it takes 9 hours to leach the base-metal chlorides of an eight-ton charge. How long it would take to leach a similar charge of the other ore I do not know, as this ore, on account of its exceedingly refractory character, has never, thus far, been treated by itself without adding other ores. But assuming the filtering property of both ores to be alike, it would follow, according to the quantity of water required, that the San Francisco del Oro ore will require 51 hours for the extraction of the base-metals by the common tank-lixiviation. While the results of these two experiments, as I said above, are not sufficient to form a conclusive opinion, they strongly indicate that in trough-lixiviation even large quantities of metal chlorides can be dissolved almost instantly, if the proper volume of water is used, an advantage which will fully be appreciated by all who have to work exceedingly refractory ores.

Silver-Leaching.

After the charge in tank D, Fig. 1, has been prepared as described above, it is sluiced out with sodium hyposulphite solution to extract the silver. The pulp enters the silver leach-trough and flows down to tank D′ of the silver department. The arrange-

ment and construction of the tanks are the same as in the base-metal department; and when operations have been started, the connections have to be set the same as described for base-metal leaching.

By using the proper proportion of solvent and ore, the latter will drop as tailings into the tank, while all the silver chloride which can be extracted by the common chlorination-test will be dissolved by the solution. The clear solution leaving the last tank H′, flows through the silver-solution trough into the distributing-box M, and from there to any desired precipitating-tank. The bottom of the distributing-box has twice as many holes as there are precipitating-tanks. In these holes bent lead pipes are inserted from below and are fastened by flanges. Two-inch hose, of which two lead to each precipitating-tank, are attached to these pipes. The holes can be closed by long plugs. This arrangement I introduced eight years ago and have found it very convenient. The operator can direct the stream from the main working-floor, without being obliged to creep over all the precipitating-tanks, as is customary in many lixiviating-works.

When one tank is filled with tailings, it is disconnected from the circuit and the pulp is admitted into the next tank. Outlet *s*′, under the filter, is opened; the solution still contained in the tank is allowed to drain off, and the part retained by absorption is displaced by water. The tailings are then sluiced out with water. Where water is scarce, the wash-water can be used for this purpose.

Proportion of Solution and Ore.

The quantity of solution to be used for a given weight of ore depends on the nature of the ore and has to be ascertained by previous experiments. Lead sulphate diminishes the solving energy of the solution for silver chloride, and therefore on account of this and some other cause, ore containing lead will require the most, while ores free from lead, but containing copper will require the least quantity of solution, because cuprous chloride increases the solving energy of the solution. In most cases the proportion of 10 weights of solution to one of ore will be sufficient. Ore from the Yedras Mine can be successfully treated with a proportion of 3 to 1. The right proportion can easily be ascertained by constructing a temporary triangular trough, about 150 feet in length, with ¾-inch fall to the foot. The upper end has to be connected with a mixing-

box 18 x 24 inches, into which a small tank of a few cubic feet discharges. The mixing-box has a partition 4 inches high and about 6 inches back from that side which contains the outlet into the trough, in order to prevent the ore from being washed down in a mass. The small tank is filled with solution to a certain mark, at which mark the surface has to be kept during the time of the experiment, by adding solution in the same proportion as it flows out. After having once ascertained the flow of solution per minute, the quantity of ore to be charged per minute has to be regulated according to the desired proportion. The samples have to be taken at the lower end of the trough, with a vessel large enough to hold the whole stream while the sample is taken. Time must be given the ore to settle. The clear solution is decanted, the ore placed on a filter, washed with water, dried and assayed. Before making the experiment with solution the base-metals have to be removed from the ore.

In the laboratory, experiments can be made on the principle of trough-lixiviation by introducing 20 grammes of roasted ore, which has previously been washed, into a graduated cylinder of 1000 c.c., in which is contained 100 c.c. or 200 c.c. of sodium hyposulphite solution, according to the proportion which is intended to be used. The top of the cylinder has to be tightly closed with the palm of the hand, and the cylinder brought into a horizontal position, and then oscillated in order to make the ore and solution pass quickly from one end to the other, to imitate the current in a trough. This is done for 1½ minute, then the contents of the cylinder are poured into a filter, washed with water to displace the silver-solution from the sand and paper, dried and assayed.

Having by experiment found the required proportion of solution and ore, the size of pumps, pipes, outlets, etc., can be calculated. Similar experiments have to be made with regard to base-metal leaching. It is also of importance to ascertain the most suitable strength of the solution.

The Troughs.

A triangular shape of the troughs is preferable, because the same quantity of water will display more energy for moving the sand than on a flat bottom. An inclination of three-fourths inch per foot is sufficient. I am not prepared yet to state the required length. With one exception, my experiments have been conducted with troughs of 150 feet in length, and the extraction was always complete

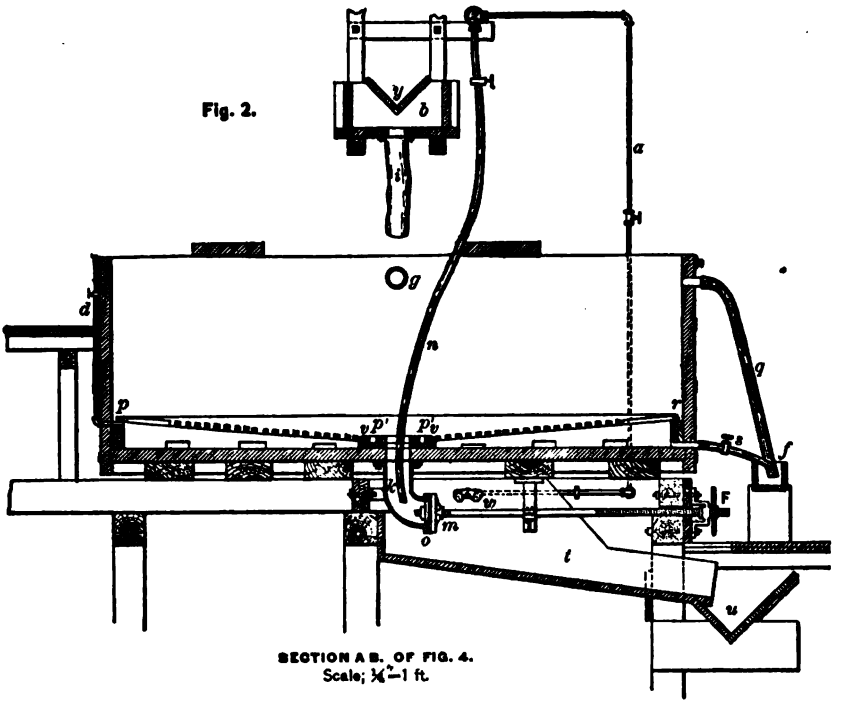
if the right proportion of solvent and ore had been used. Therefore I can at present only say that 150 feet will be sufficient in most cases. It did not occur to me before to try shorter troughs with the view of ascertaining the shortest length, because I naturally was of the opinion that the longer the trough the better the effect; but now I am strongly inclined to believe that much shorter troughs will be of equally good effect, at least with regard to silver-leaching, for the following reasons: At the North Mexican Mill, for want of boards I used pipes, 150 feet from the hopper or mixing-box to the first settling-tank. On account of bends and intersections it took the pulp 1¼ minute to pass through. The extraction was complete, provided the right proportion was used. At Yedras, where I was kindly permitted to make a few experiments for my own information, I had a straight trough 150 feet long and ¾ inch fall per foot. The pulp rushed through in ¾ of a minute and the extraction was complete. Now, if the trough at Yedras had had bends and intersections, and thus the pulp had moved with the same velocity as in the North Mexican Mill, it would have taken but 90 feet of trough to retain the pulp ¾ of a minute, and to perform the extraction.

My first experimental trough was only 43 feet long. I tried some ore from the Cusi mine; but not knowing at that time the importance of maintaining certain proportions, I used 6 : 1, which I afterwards found not to be sufficient for this ore. The result, therefore, was not satisfactory. By maintaining the same proportion, but letting the pulp pass twice through the trough, or 86 feet, the result was exactly the same. By passing three times, or 129 feet, the extraction was only a fraction of an ounce better. This shows that 43 feet of trough had the same effect as 86 or 129 feet. I will investigate this important point at the first opportunity I have to experiment again on a large scale. If short troughs will answer, much less grade will be required. At all events, it is an easy matter to ascertain the required length of both the base-metal and the silver-leach troughs before erecting the works.

Sluice-Tanks and Sluicing.

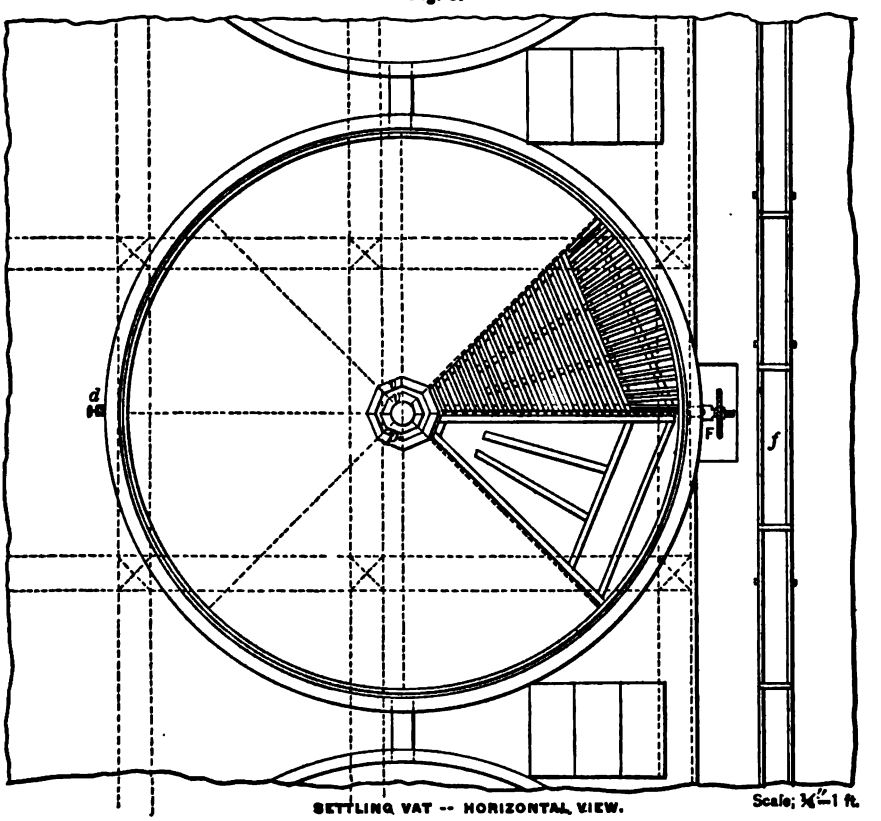
Fig. 2 represents the vertical section, Fig. 3 the horizontal view, and Fig. 4 the front view of a settling-tank arranged for sluicing. In the centre of the bottom is the discharge-opening, 6 inches in diameter. The cast-iron discharge-tube *k*, of the same inside diameter, tightly fastened to the outside of the tank-bottom, corre-

Fig. 2.



SECTION A B. OF FIG. 4.
Scale; ½"=1 ft.

Fig. 3.



SETTLING VAT -- HORIZONTAL VIEW.

Scale; ½"=1 ft.

sponds with the discharge-hole. The lower end of the tube is at right-angles with the upper end, and provided with flange *o*. The valve *m*, which is provided with a rubber gasket, can be pressed tightly against flange *o* by turning the wheel F. Flange *o* and valve *m* are made of brass. Fig. 4*a* shows in detail the construction of wheel F. Around the discharge-opening, and fastened to the bottom of the tank, is the wooden polygon *v*, in which is cut the groove *p'*. Around the inner periphery of the tank, and high enough to give the filter-bottom an inclination of at least $\frac{3}{4}$ inch to the foot, is the groove *p*. Fig. 3 illustrates the construction of the filter-bottom, which is made in section. The filter-cloth is well fastened, and kept in position by driving tightly a rope into the grooves *p'* and *p*. The air-escape pipe *d*, which reaches to the rim of the tank, enters the latter close under the filter-bottom. A piece of hose is fastened to the upper end, and can be closed by a hose-clamp. In Fig. 4 *q q* are solution-outlets; *s*, filter-outlet. Connecting pipes *g* and *h* (Figs. 2 and 4) have, like the discharge-tube *k*, to be well coated with asphaltum varnish. In the same figures, *z* is the water-pipe; *n* the central hose, which reaches down into the discharge-tube *k*, where it has to remain during the process of charging. This hose ought to be very stiff. Before charging the tank the discharge-pipe is filled with water through the central hose, in order to keep the latter filled with water, which will prevent the inside of the hose from being obstructed by ore. In the base-metal department the central hose has to be connected with the water- and solution-pipe, to allow the application of either, while in the silver department they need only to be connected with the water-pipe.

When a tank is ready to be discharged, the wheel F is turned, and thus the valve *m* pulled back. The water is injected through the central hose, while the latter is gently moved up and down. The stream undermines the tightly-packed sand, and causes a continual caving-in, until a funnel-shaped opening is made through its depth to the surface. Then several streams are made to play on the top, while the central hose, with checked stream, is left in position to avoid obstruction of the discharge-pipe by a too sudden rush of sand.

The central position of the discharge-opening and the funnel-shape of the filter permit a quick and clean sluicing. The pulp leaving the discharge-tube enters the sluice-trough *t* underneath the tank, which leads to the tailings-trough *u* in the silver department, or to the silver-leach trough in the base-metal department. The charge being sluiced out, tank and filter have to be well rinsed, the

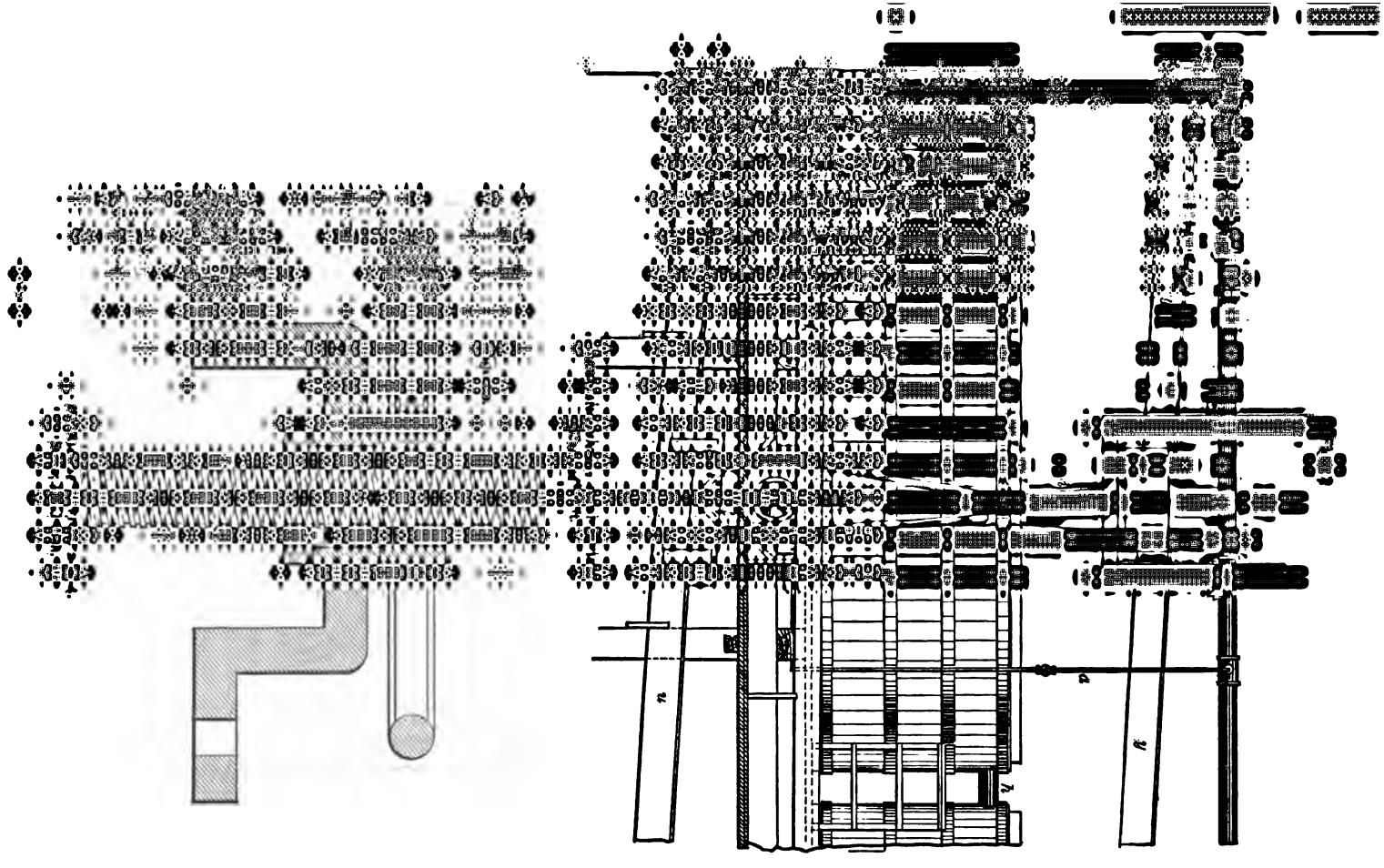
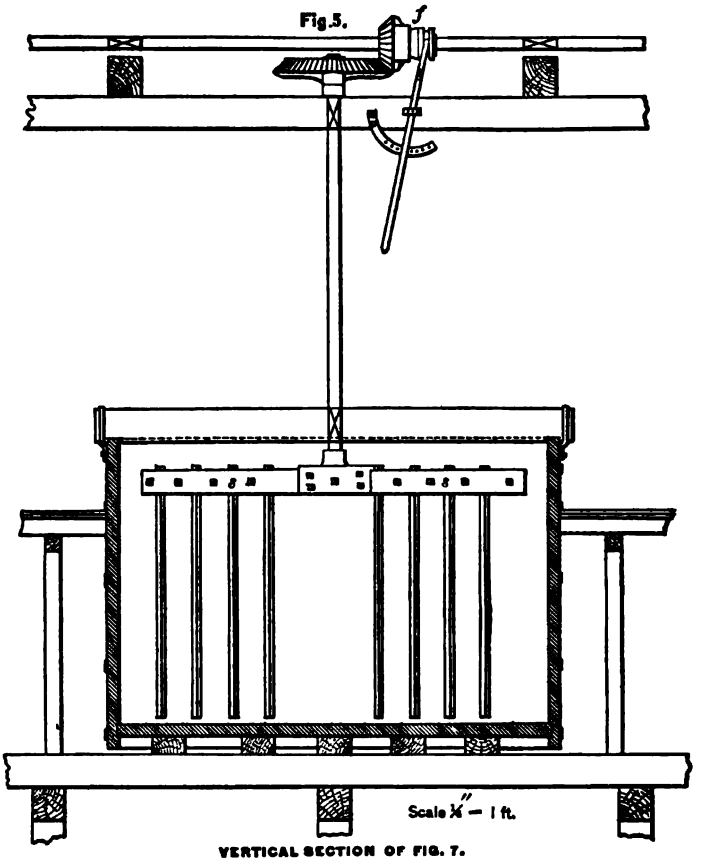
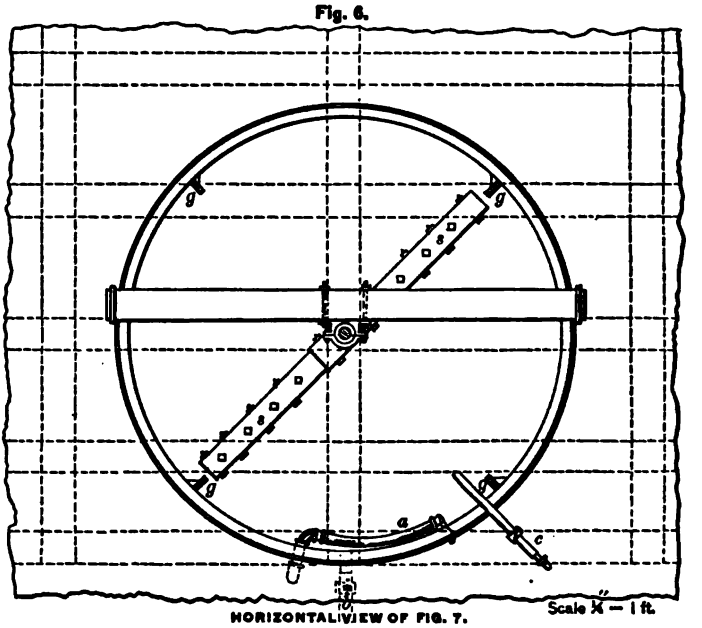


Fig. 5.



VERTICAL SECTION OF FIG. 7.

Fig. 6.



HORIZONTAL VIEW OF FIG. 7.

valve of branch-pipe *a* is opened, and the adhering sand washed off from flange *o* and valve *m* by the double sprinkler *w*. Then the discharge-valve is closed again, and the tank is ready to be connected with the other tanks. In Figs. 2 and 4 *y* represents the silver-leach trough; *b*, the intersecting-box above the tank, and *i*, a short hose made of duck.

Precipitating-Vat.

Figs. 5 to 9 illustrate the construction of my precipitating-vats, and also a very convenient arrangement of filters for the precipitate. The tanks are provided with a machine-stirrer of the construction indicated in the drawings. The stirrer *s* (Figs. 5 and 6) has to make about 30 revolutions per minute if the diameter of the tank is not more than 8 or 9 feet. It is set in motion, or stopped, by working the friction-clutch *f* (Fig. 5). The wings *g* (Fig. 6), which reach near to the bottom, are about 3 inches wide, and are kept in position by triangular pieces of boards. They break the violent current around the periphery and throw the solution towards the center, thus causing a strong whirling motion. In Figs. 6 and 7 *a* is the discharge-pipe for the clear solution, and *b* for the precipitate. Pipe *d* passes in front of all the precipitating-tanks, and conveys the calcium or sodium sulphide solution from the reservoir to the tanks. The branch-pipe *c* reaches above the rim of the tank, and ends in a rubber hose, which is provided with a clamp. In precipitating, the stream of calcium sulphide can be regulated, and the operator, by observing the color produced by the precipitant in the moving solution, can, after some practice, finish this operation in a very short time, and much more easily than by using buckets.

I introduced precipitating-vats with machine-stirrer first at the North Mexican mill in Cusihuiriachic, Mex., where they gave much satisfaction. One man could precipitate three tanks at a time, without assistance. The solution was so thoroughly agitated, that a very quick and perfect separation of the silver sulphide took place. The separation was so perfect that the bottom of the tank could distinctly be seen through 5 feet of solution.

In most of the leaching-works it is customary to erect only two or three very large precipitating-vats, which, in my opinion, is not favorable to precipitation. To produce a quick and perfect separation of the precipitated silver sulphide, the solution has to be vigorously agitated. This cannot be done in tanks of 14 or 16 feet in diameter. It is much preferable to have smaller tanks and a larger number of them. I construct them 8 to 9 feet in diameter and 6

feet deep. Another bad practice which I have noticed in some leaching-works is to discharge the precipitate only once a week, in some even only once a month. Fresh precipitate forms large flakes which settle easily. After two or three days it assumes a dry and sandy condition, and if stirred up, divides into very fine particles, which are kept suspended in the solution for a long time. The result of such a practice is, that the decanted solution will not be free from precipitate when used again for extraction, and a black coating of precipitate will cover the surface of the ore in the leaching tanks. This black coating is usually much poorer than the balance of the precipitate, but it is always too valuable to be lost. The precipitate ought therefore to be removed from the precipitating-tanks every day, and should never be allowed to remain longer than two days. By using smaller tanks with machine-stirrer, and by discharging the precipitate every or every other day, the circulating solution can be kept so clear, that even after a prolonged leaching of five or six days no black coating can be observed on the top of the ore.

The machine-stirrer does also good work in discharging the precipitate. After the clear solution has been decanted within a few inches of the precipitate, the stirrer is set in motion, and the discharge-pipe *b* opened. The stirrer agitates the precipitate until nearly all is discharged into the filters or into the vat from which the filter-press is supplied. Thus the cleaning of a tank can be done in a very short time.

Where the filter-press is not used, a filter-arrangement as shown in Figs. 7, 8 and 9, will be found very convenient. Two rows of filters are so arranged that they are in commuication with each other by depressions cut into the divides of the frame. By allowing the precipitate to flow into one filter, all the adjoining filters will gradually be filled one after the other, without requiring the attention of the operator. The filters are shallow, and the precipitate will thus be spread in a comparatively thin layer over a large filtering-surface. Under each row of filters is placed a trough, *n*, which receives the filtrate and conveys it to the pump-tanks, *P*, below the floor. These filter-frames are placed in front of the precipitating-tanks and can be made to contain quite a number of filters, as shown in Fig. 1, where two sections serve for six precipitating-tanks. Owing to the shallowness of the filters and the large filtering-surface, the solution will drain off fast, and in five or six hours the precipitate will be stiff enough to be charged with a hand-shovel into the furnace.

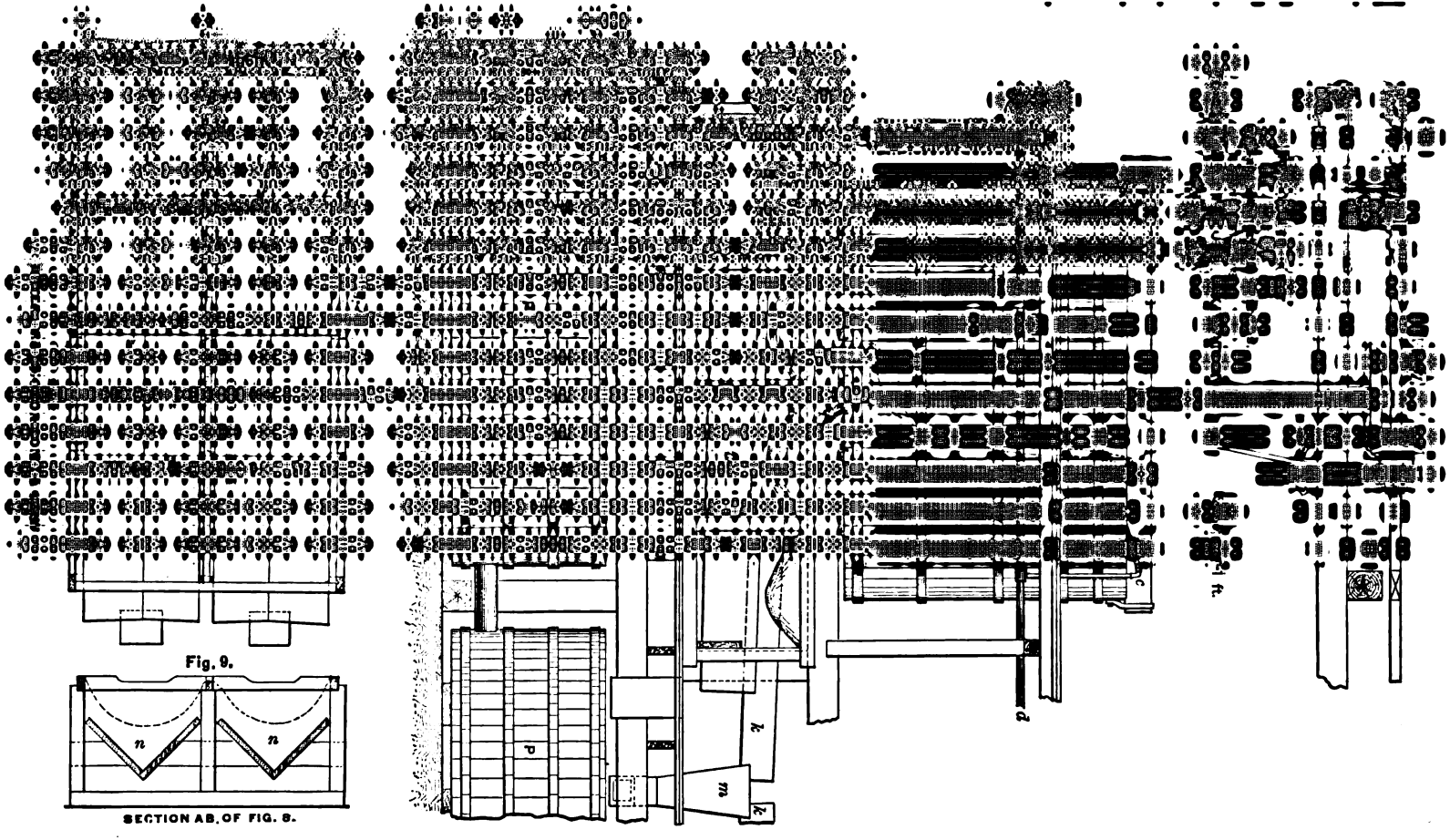


Fig. 9.

SECTION AB, OF FIG. 8.

Fig. 10.

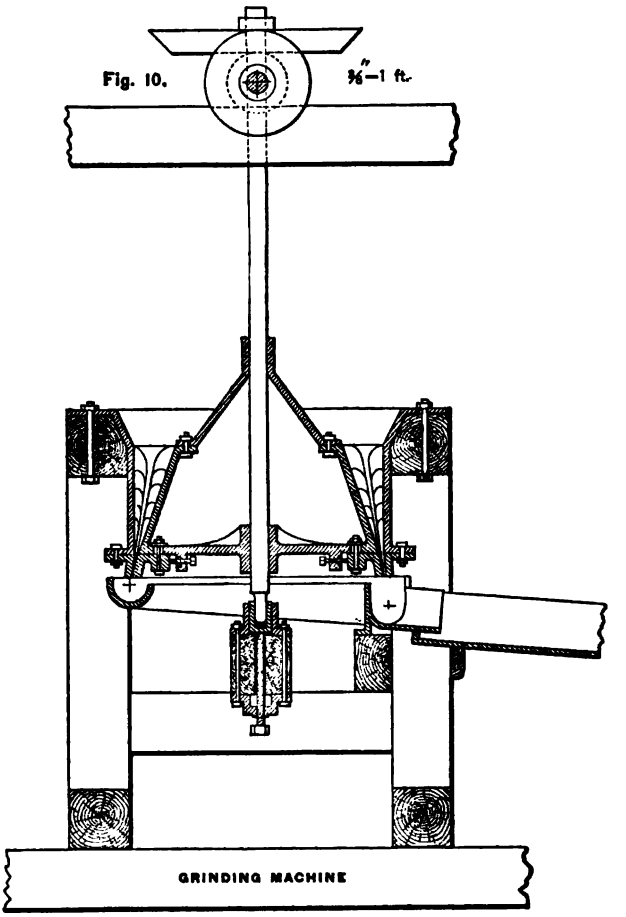
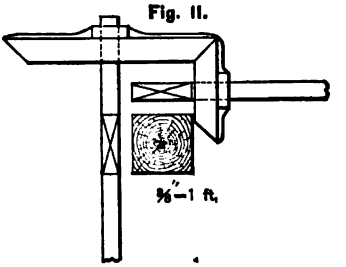


Fig. 11.

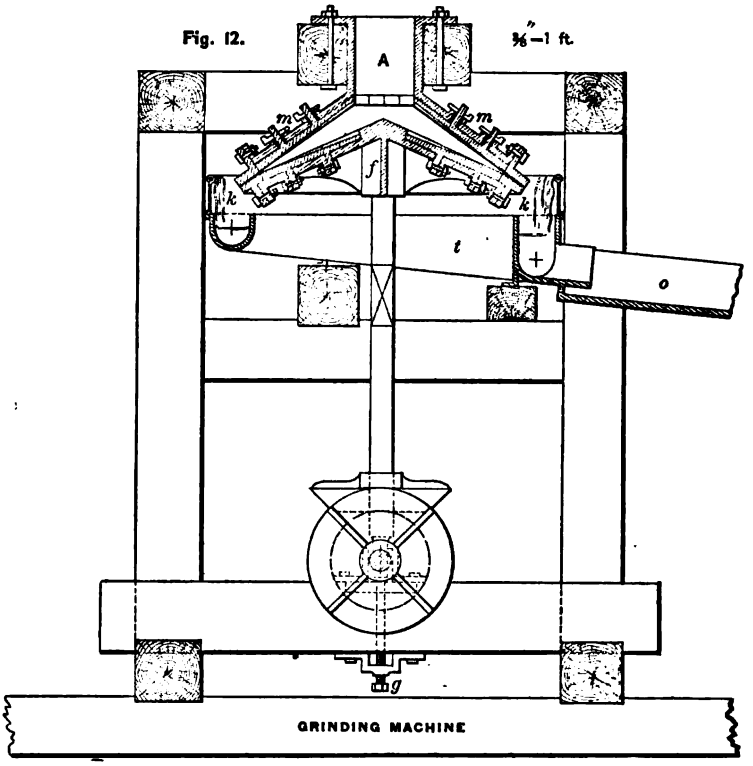


VERTICAL SECTION THROUGH UPPER PART OF FIG. 10.

Crushing the Lumps of Roasted Ore.

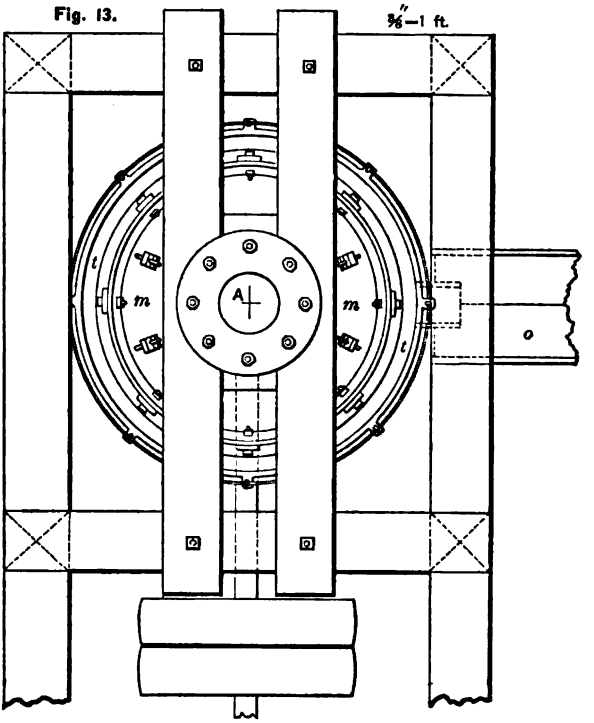
The extraction in trough-lixiviation is not produced by filtration; and the silver as well as the base-metal chlorides contained in the lumps of the roasted ore will not be extracted if the lumps are allowed to enter the trough together with the finer portion of the ore. In order to obtain good results it is necessary first to mash the lumps. An agitator will not perform the work rapidly enough, and some quick-grinding machine has to be employed for this purpose. Figs. 10 and 11 show the construction of the *Kegelmühle*, which is very effective on soft material, and is used in Europe for grinding coal. It is copied from Rittinger, with some slight alterations, which I made to adapt it to trough-lixiviation. Figs. 12, 13, 14 and 15 represent another grinding-machine for soft material, which is my own device. It is based on the principle of the *Kegelmühle*. The mantle *m* is also stationary, but, instead of being cylindrical, is cone-shaped. The muller *f*, a flatter cone than the mantle, is inserted from below, and can be lowered or raised by the screw *g*, for the purpose of regulating the fineness to which the lumps are intended to be mashed. Mantle and muller are provided with exchangeable shoes and dies. This mill has a larger working capacity than the *Kegelmühle*, because the centrifugal force, created by the rotation of the muller, will greatly assist the discharge of the pulp, while in the *Kegelmühle*, with an almost vertical discharge, the centrifugal force has no beneficial effect, but on the contrary, acts against the full discharge of the pulp. Towards the center, shoes and dies are provided with teeth, while towards the periphery their surface is smooth. The teeth cut the larger lumps, the smaller ones are mashed by the smooth part of the cones. Water and ore are charged through *A*, while the circular cast-iron trough *t* receives the pulp and conveys it to the base metal leach-trough *o*. The canvas strip *k* prevents the pulp from flying over the rim of the trough. The main portion of the roasted ore will be fine enough to pass through the mill without being affected, and only a small part will have to be mashed. For this reason, and on account of the softness of the wet material, large quantities of ore can be put through in twenty-four hours with but very little wear of the machine, which, in itself, is of simple and cheap construction.

Fig. 12. $\frac{3}{8}$"—1 ft.



GRINDING MACHINE

Fig. 13. $\frac{3}{8}$"—1 ft.



TOP VIEW OF FIG. 12.

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Fig. 15.

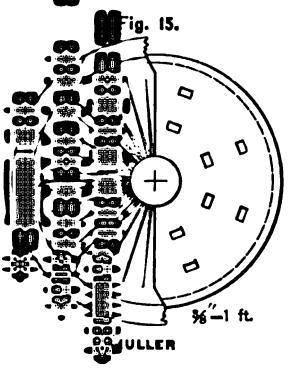
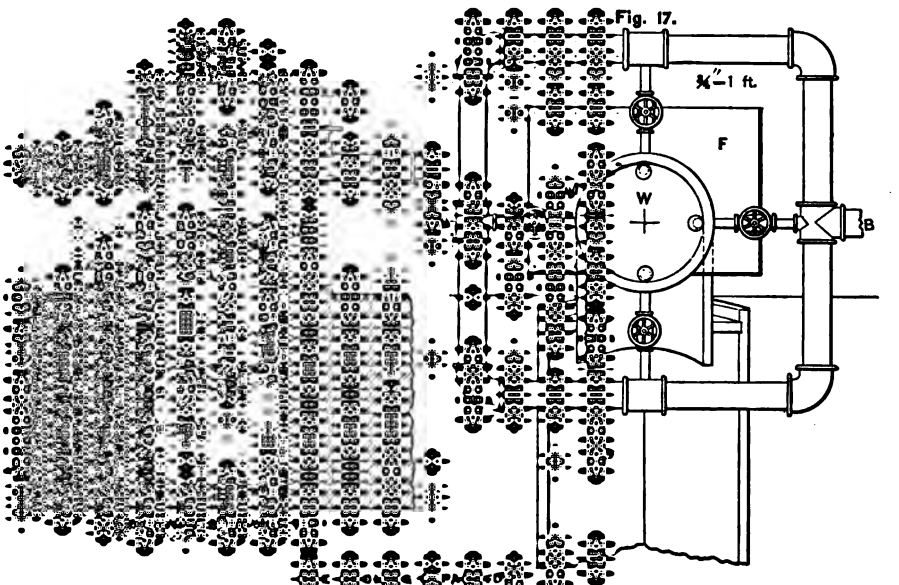


Fig. 17.



resented in Figs. 16
a flange or plate F
of the tube has the
ugh O, which leads

to the grinding-machine. Inside the tube, and on two different levels, water-spouts are so arranged and constructed that the entering water forms sheets (not sprays) which perfectly close the tube. At the line where the two opposite sheets meet, the water will drop straight down. The sheets of water are produced by placing a smooth plate of iron properly over the obliquely-cut ends of the pipes, as shown in Fig. 16. The water-pipes inside the cooler ought to be made of cast-iron, to resist longer the action of the solution. The hot ore drops through W, and, in striking the water, is immediately carried below the sheets of water and is so quickly enwrapped in it that but little dust is formed, which will be absorbed by the sprays, while most of the steam will be condensed. The bottom T has sufficient inclination to make the ore move as soon as it strikes it.

By cooling the ore in this way, we have the additional advantage of increasing the temperature of the water used for base-metal leaching, without incurring extra expense or labor. The resulting temperature of the water will depend on its original temperature, the temperature of the ore and the proportion of water and ore used. By a laboratory-test I found that a dark red-hot ore, dropped in three times its weight of water, raises the temperature of the latter from 60° F. to 95° F.

Hot-Ore Feeding-Machine.

In order to work the cooling-apparatus to advantage, it is necessary to charge the hot ore evenly by some mechanical contrivance. I have devised an arrangement which, in combination with a Howell roasting-furnace, is illustrated in Figs. 18, 19 and 20. The ore from the furnace drops alternately into the bins or vaults, E and F. The ore is removed from the bins by conveyer-screws. These, with the conveyer-troughs, are placed below the inclined bottom of the bins in order to keep them comparatively cool by the air in the channels, *f*. The 4-inch shaft of the conveyer, over which the cast-iron screw-segments are slipped, consists of three pieces which are connected close in front of the journals by large flanges, *g*, in such a way that the coupled ends do not touch each other. The object of these flanges is to keep the journals cool. For this purpose the vessel, *h*, through which a small but continuous stream of water flows, is so placed, that the flanges have to revolve through the water. The inflowing water strikes the upper half of the couplings to keep them

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[illegible] diagram, whence the
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HOT-ORE FEEDER--VERTICAL SECTION.
Scale: ½" = 1 ft.

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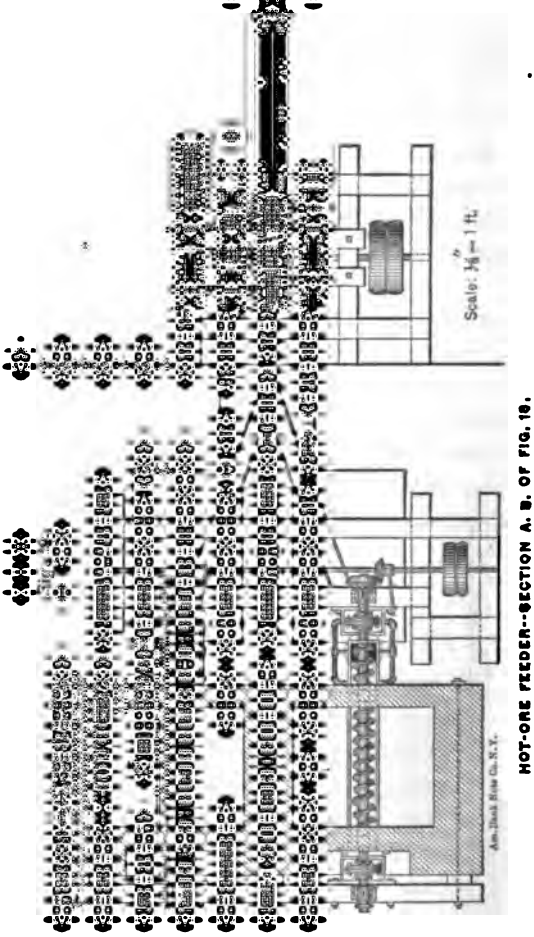

HOT-ORE FEEDER--SECTION A. B. OF FIG. 18.

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ribed arrangement impracti-

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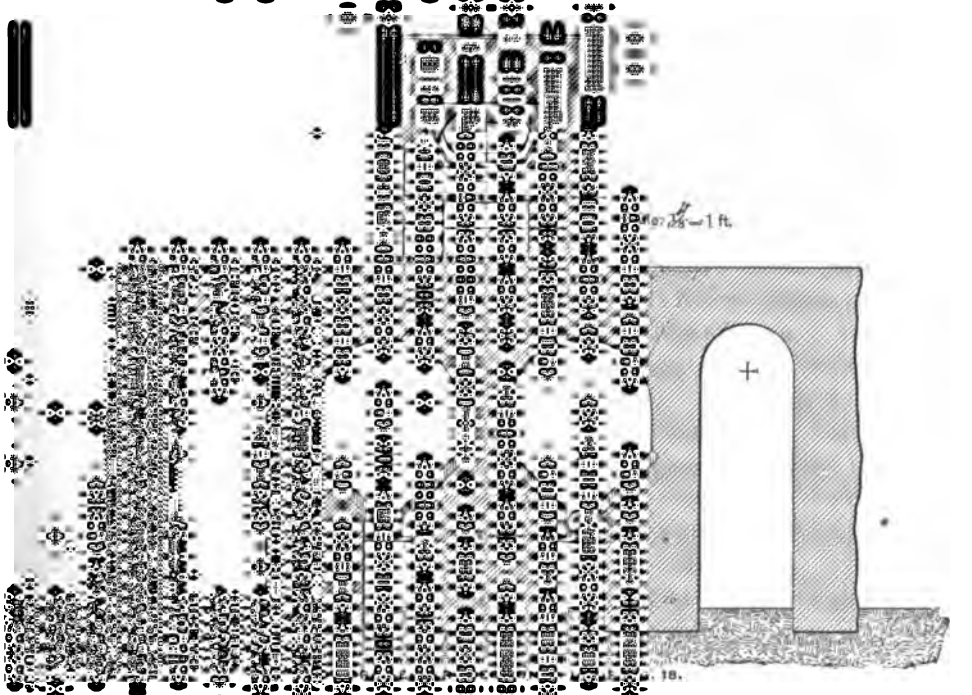

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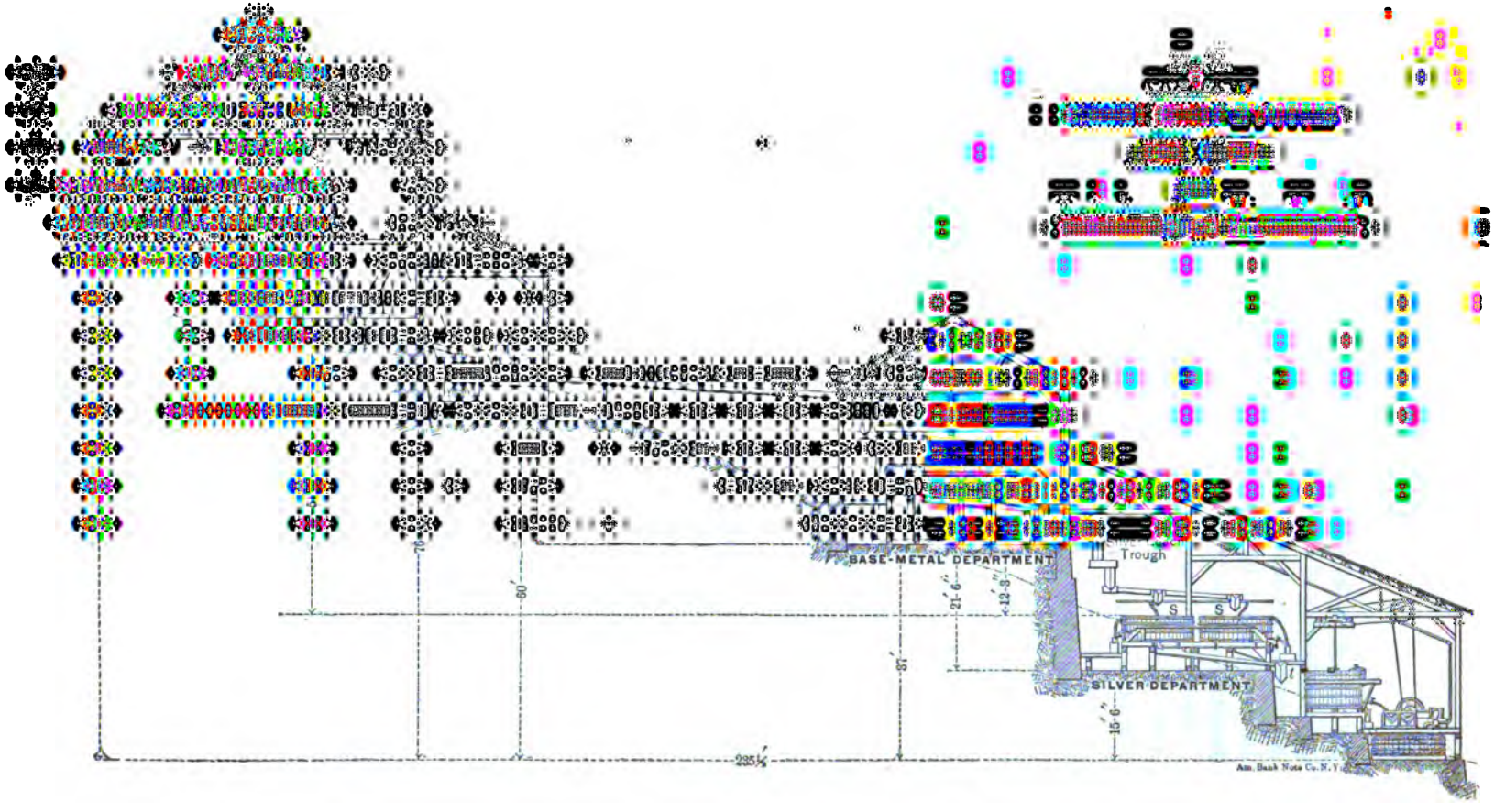
I do not intend to represent the above hot-ore feeding-machines for refractory ores as perfect; but, with some manual assistance, they will perform the work, and will give much more satisfaction than the present cooling-floor manipulations.

Complete Trough-Lixiviating Works.

If circumstances require it, trough-lixiviation can be used only for the extraction of the silver, while the base metals can be removed in the usual way. This may be necessary in case tank-lixiviating works have to be altered into trough-lixiviating works. In such cases, it will require only a few base-metal leaching-tanks to be erected in or outside the main building, at a place which offers sufficient grade for the silver-leach trough. It is immaterial where these tanks are placed, as the connection with the silver department can easily be made by the silver-leach trough. The roasted ore will have to be elevated from the cooling-floor to the level of the base-metal tanks. There are ores which retain the silver chloride tenaciously, and require a very long leaching-time. In some works five or six days, and even more, is not unusual. By changing such works into trough-lixiviation, this annoying difficulty would be entirely overcome at comparatively small expense.

If the entire works are arranged for trough-lixiviation, the advantages derived from the new system are, of course, much greater. Fig. 21 shows the complete system in connection with a Howell furnace. The leaching-works are not in the same building as the battery and furnaces, but some distance down the slope of the site, in order to gain grade. Hot-ore feeder, ore-cooling apparatus and grinding-machine are in the furnace-building. The ore, after passing through the furnace, drops into the bin, F, from which it is removed after some hours by the conveyer, and drops through the cooler. From here the water carries it through trough *k*, to the grinding machine. After passing through the grinder, the pulp enters the base-metal leach-trough, which conveys it down to the base-metal department, and drops into the settling-tanks, B. While the ore passes through the trough, the base metals are extracted. From here the ore is sluiced out with sodium hyposulphite solution into the silver-leach trough, and conveyed to the settling-tanks, S, in the silver-department. The extraction of the silver-chloride being performed in the trough, the sand drops into these tanks as tailings. The silver-solution flows into the precipitating-tanks, and the tailings are sluiced out into tailing-trough, *t*. The leaching-troughs have ¼-inch fall


BASE-METAL DEPARTMENT
Trough
SILVER DEPARTMENT
Am. Bank Note Co. N.Y.

per foot, with the exception of trough *k*, in the furnace-building, which has more (about 2¾ inches per foot), because the material to be moved in it is coarse. This trough is not triangular, like the others, but has a round bottom, in order to give the water a better opportunity to move larger lumps.

One of the objections used against trough-lixiviation has been, that a mill with this system will require more fall then an ordinary mill. Though this is true, I cannot see why this should be an obstacle to the new system. The additional grade is not needed as a sudden drop in one particular place, but is divided over the entire works; and by erecting the leaching-works further down the site, no unusual steep slope is required. As a precaution against a total destruction of the works by fire, it is even advisable not to bring all the departments under one roof, especially in this case, where the transfer of the ore from one department to the other is performed automatically and without expense.

In order to illustrate that the extra grade required does not offer any difficulties in the construction of the buildings, and that the trough-system does not require any unusually steep slope, or unusually large excavations, I have constructed the plant as represented in Figs. 1 and 21, with excessively long troughs:—base-metal leach-trough from cooler to settling-tank, E, in the base-metal department, 293 feet, and silver-leach trough from tank A (base-metal department) to tank E′ in silver-department (Fig. 1) 344 feet; in all 637 feet. It can be seen at a glance that the plant can be erected without any difficulties, and that no unusually large excavations are required. The slope of the site as represented is 22°, which I do not think an unusual grade for any mining district.

Advantages of Trough-Lixiviation.

I will now point out, in a few words, the advantages of the new system.

1. The system being continuous, the operator receives for precipitation a silver solution of uniform strength in silver, which facilitates quick and exact precipitation.

2. In tank-lixiviation, each tank-charge is in a different stage of the process, which requires a great deal of attention, and offers the opportunity for costly mistakes. In smaller works, this is not so

much felt, but in larger works like the Cusi mill, where twenty-two leaching-tanks are in operation, and each of them in a different stage of treatment, it becomes quite complicated. Notwithstanding the trough-system requires two departments, the operations are materially simplified. In each department, attention has to be paid only to about two tanks, and the operator will have plenty of time left to attend to precipitation, etc.

3. The most important advantage of trough-lixiviation is the fact that this method enables the operator to bring the ore into sudden contact with any desired quantity of the solvent. This is a very important fact, as it offers the means to overcome some annoying chemical and mechanical difficulties, hitherto encountered in the lixiviating process. The extraction of the silver from lead-bearing ores is slow, and requires an extensive plant. It is well-known to leachers of such ores that, while the main portion of the silver is extracted in a short time, the remaining few ounces will be tenaciously retained by the ore. Thus it happens that while the main portion of the silver was extracted in the first six or eight hours, the remaining eight or ten ounces of extractable silver will require three or four days, sometimes more. I have noticed cases in which only one ounce per ton in every twenty-four hours of prolonged lixiviation could be extracted. This very singular phenomenon is difficult to explain. It seems that only that part of the silver is so difficult to extract which originally was contained in the lead-mineral of the ore. I have observed that whenever the galena of the ore became richer in silver or had increased in quantity, the extraction became slow and dragging. This, together with the fact that the main portion of the silver can be quickly extracted, indicates that the slowness of the extraction is not principally due to the disadvantageous influence of lead sulphate on the dissolving energy of the solution for silver chloride, but that it must be due to some other cause which makes the silver contained in the lead-ore to dissolve only in large volumes of sodium hyposulphite solution. In troughs, I can bring the ore at once into contact with the required volume of solution, and thus produce almost instantaneous extraction. I have made experiments with some low-grade and rather "leady" ore from the Cusi mine:

Roasted ore, ...	27.12 ounces silver per ton
Chlorination-test, tailings, . . .	3.94 " "

By treating it on the trough-principle with different quantities of solution, I obtained the following results:

6	weights of solution to 1 of ore,	.	.	Tailings,	14.58	ounces	per	T.
8	"	"	"	.	.	"	6.56	"	"
12	"	"	"	.	.	"	5.25	"	"
18	"	"	"	.	.	"	4.37	"	"
24	"	"	"	.	.	"	4.37	"	"

While these results show that the proportion of 18 : 1 is sufficient, we can safely assume that less solution would have had the same effect, say 14 : 1. According to the filtering-property of the Cusi ore, it would take seventy hours to bring the ore in contact with fourteen times its weight, a leaching time which is not infrequent with ore from certain parts of the mine.

The presence of lead does not retard trough-lixiviation: it merely entails the use of larger quantities of solution, which is surely a great advantage.

The rapidity with which the silver chloride dissolves, also materially lessens the unfavorable influence of caustic lime.

4. This system is especially adaptable to larger works, since large quantities of ore can be worked with a comparatively small plant.

5. By producing at once a sufficiently diluted base-metal solution, the small quantities of silver chloride which have dissolved will be precipitated, and will settle in the tanks; and thus the main part of it can be regained without a separate treatment.

6. The system is almost entirely automatic, and offers the possibility of avoiding the costly cooling-floor manipulations which cannot be accomplished in tank-lixiviation. As, besides this, neither the charging nor the discharging of the tanks has to be done by manual labor, it is apparent that trough-lixiviation saves a great deal of manual labor without substituting for it much machinery, and thus reduces materially the current expenses. Even large trough-lixiviating works can be operated with only a few hands, an advantage which will be greatly appreciated by managers of mining enterprises in Mexico, where the inconstancy of the common laborer causes irregularity and great annoyance.

SOME RECENT IMPROVEMENTS IN OPEN-HEARTH STEEL PRACTICE.

BY ALFRED E. HUNT, PITTSBURGH, PA.

(Boston Meeting, February, 1888.)

THE late Alexander Holley said, on returning from a careful study of the relative merits of the Bessemer and the open-hearth processes, as shown in the best European practice, that, in this country, the open-hearth process would live to attend the funeral of the Bessemer; and he said this several years ago, when in this country the Bessemer plant and practice had been developed to far outstrip that abroad, and when, proportionately, it had reached an advancement much beyond the improvements in open-hearth plants and practice. At the same time, the prevailing idea in the metallurgical world, a few years ago, seemed to be in favor of the substitution of Bessemer for open-hearth steel for most of the higher qualities, and Bessemer enthusiasts loudly proclaimed, that as good steel for all purposes could be obtained by their favorite process as by the open-hearth. It was, in fact, confidently prophesied that very soon tool-steel and similar grades, which consumers had learned to specify, in ordering, to be made by the crucible process, would by common consent be made by the Bessemer, as best adapted to give the desired results. But these expectations have failed to materialize; not only do the crucible people keep on in the even tenor of their way and talk with pride of their comparatively large outputs of "hundreds" of tons per month, as usual, but the open-hearth people have kept fast hold of their original domain of nearly the entire amount of sheared plates, of locomotive tires, of large forgings, spring-steel, agricultural steel, steel castings and the like. Further, owing largely to one of the great advantages of the Bessemer process, which is also one of its inherent dangers, namely, its large output, it has also fallen into considerable disrepute for structural purposes. Not a few experienced engineers are now stipulating in their specifications that Bessemer steel will not be allowed to be used, especially for tension-members.

Quite a number of the Bessemer works now have structural trains in addition to their older rail- and bloom-mills, both of which the managers desire to keep in constant operation. To accomplish this end they have charged alternately in their cupolas mixtures of pig-iron suitable for making rail-steel and also the higher qualities suitable for structural purposes. Occasionally these various qualities of pig metal get mixed during the melting operation.

The working-force at the pits, the blooming-mills and in the bloom-yards is trained to get the steel out of the way as rapidly as possible, often regardless of the precise quality which may have been delivered from the converters. The careful manipulation and inspection needful to keep the metal from getting mixed and insure the rejection of poor or defective steel, is very difficult in the rapid handling of such large quantities as the Bessemer process turns out both day and night.

An important and well-recognized advantage in the making of uniformly good steel by the open-hearth over the Bessemer process is the length of time available for examining and testing the quality of the metal after it is ready to tap, by dipping out small ladlefuls of steel from various parts of the bath and casting into small test-ingots. Too rapid rolling, too much draught between successive passages through the rolls, and finishing of the rolling at too high a heat are causes of failure in steel, which have proved much more likely to occur at Bessemer mills, where all hands are trained for output and tonnage, than at the open-hearth blooming-mills, where the men are more conservative and slower in their work and are less pushed.

These remarks are obviously general in their character. The writer has seen some of the best and most homogeneous tough and ductile steel ever made, produced by the Bessemer process; but for uniformity and homogeneity, together with the best results for tenacity and ductility, it is conceded by nearly everyone that the slower-working open-hearth process still retains the supremacy over the Bessemer which it at first won for itself. J. W. Wailes, in his paper, read before the Chemical Section of the British Association, in 1887, says: "Periodically the users of soft steel receive a shock —a mysterious failure is reported—a bar or plate has cracked like a sheet of glass; all the chemical analyses show pure metal. It has been general experience that it was revealed on investigation that these failures occur in steel of one class, viz., soft steel made by the

Bessemer process, steel made where the process had been pushed to its utmost limit and oxidation had been greatest." In soft steel, where the operation of oxidation of the carbon has to be carried to an extreme limit, and a copious supply of oxidizing medium is provided, it is better to supply this oxidizing medium from ore in its natural state and unreduced condition rather than to reoxidize iron that has been already reduced at cost of fuel and labor. The extreme violence of the oxidation by pneumatic means renders it more difficult to control it regularly. The writer does not dispute that so far as ordinary chemical analysis of each heat or blow indicates its regular chemical composition, the Bessemer has obtained as much uniformity as the open-hearth; but the ordinary chemical analyses are not the only criteria of regularity of product. For instance, if we could get correct data as to the percentage of contained oxygen in the steel, we would probably find a considerable difference in ordinary working between the Bessemer and the open-hearth process.

A glance at the statistics given in the Iron and Steel Directory will show that the laws of supply and demand have very closely followed this popular judgment of the metallurgical values of the open-hearth and Bessemer processes. In 1880, there were 22 completed open-hearth steel-works and 3 building, with 41 open-hearth furnaces, and 11 completed Bessemer works and 2 building, with 32 converters; while in 1887 there were 50 completed open-hearth works and 3 building, with 104 open-hearth furnaces, and 35 Bessemer works with 75 converters, and at present, in February, 1888, so far as is known to the writer, there are 3 Bessemer steel plants building and 6 open-hearth plants, while of Clapp-Griffith plants, of which there were 8 built between 1884 and 1886, there are none now being built.

I. Improvements in Plant.

One of the chief defects of the original open-hearth furnace was that a large portion of the superincumbent weight of the furnace and its charge was supported by the brick walls, between the gas- and air-regenerators, which were, at the upper part of the regenerators, softened by being subjected often to intense heat on both sides. This fault is obviated, in nearly all of the newer designed furnaces, by encasing the hearth in a plate-iron shell. This shell is

carried on iron beams extending completely across the furnace and resting upon exterior walls or columns which are independent of the more highly heated and perishable parts of the furnace underneath.

Natural gas, in the favored regions, has been a great boon to the open-hearth-steel manager; he no longer has to spend a very valuable portion of his time "poking the gas-man to poke his fires." The gas is carried to the furnace in an even flow through a three-inch gas-pipe, which branches off to both ends of the furnace in two-inch pipes. The delivery and reversing of the gas are regulated by ordinary globe gas-valves placed in the circuit. The ends of the pipe are encased in the brick-work, and open into the flats of each end of the furnace from the opposite side-walls by leaving out the space of a header in the fire-brick at the ends of the pipe. This gaseous fuel is not diluted to fully 60 per cent. of its entire bulk with inert nitrogen, as is Siemens gas, which has to be conducted in pipes of four feet diameter to one of the gas-regenerators to be preheated. Natural gas is conducted directly to the ports of the furnace, as it was found that preheating decomposed it and soon filled the checker-work of the gas-regenerators with deposited carbon. The use of cold gas is much more than compensated for in the heat produced by the combustion of the concentrated fuel, and by avoiding the large amount of nitrogen which is present in Siemens gas. Natural-gas melting-furnaces are now built so that both of the regenerators at each end of the furnace are connected with the air-inlet valve, or they are built with only one regenerator at each end of the furnace for preheating the air. In this case the air-valve and air-regenerators are built larger, with 60 per cent. greater capacity than when using Siemens gas. Natural gas is supplied in the service-mains to melting-furnaces in the vicinity of Pittsburgh with a pressure of about eight ounces; and this pressure is adjusted in the service-pipes by regulators, ordinarily furnished to the plants by the natural-gas companies, and placed with their connection to their mains at some suitable point about the plant. The pressure, as regulated by the valves at the large Lash furnaces, is at present only about one ounce or an inch and a half of water-pressure, as the gas goes into the melting furnace. The lighter the pressure, so long as the flow is kept uniform and steady, the better the combustion, and the more intense the heat. The steady uniform flow of natural gas has made it peculiarly applicable as a fuel for melt-

ing in open-hearth furnaces and has been a very potent cause of the growing reputation for regularity and uniformity, as well as for superiority in other ways, of the open-hearth steel made in the natural-gas districts.

The tendency up to within the past year has been to increase the size and capacity of open-hearth steel-furnaces, and the newer furnaces have been of thirty and forty tons capacity, instead of five and seven tons, as were those built at first. Except for special purposes, where large steel castings are to be made, the writer believes the limit of size has been reached. Experience with the large furnaces has led to the growing conviction at present that furnaces of about 15 tons to 20 tons, at the maximum, are the most economical and produce the best steel.

The Lash Furnace.

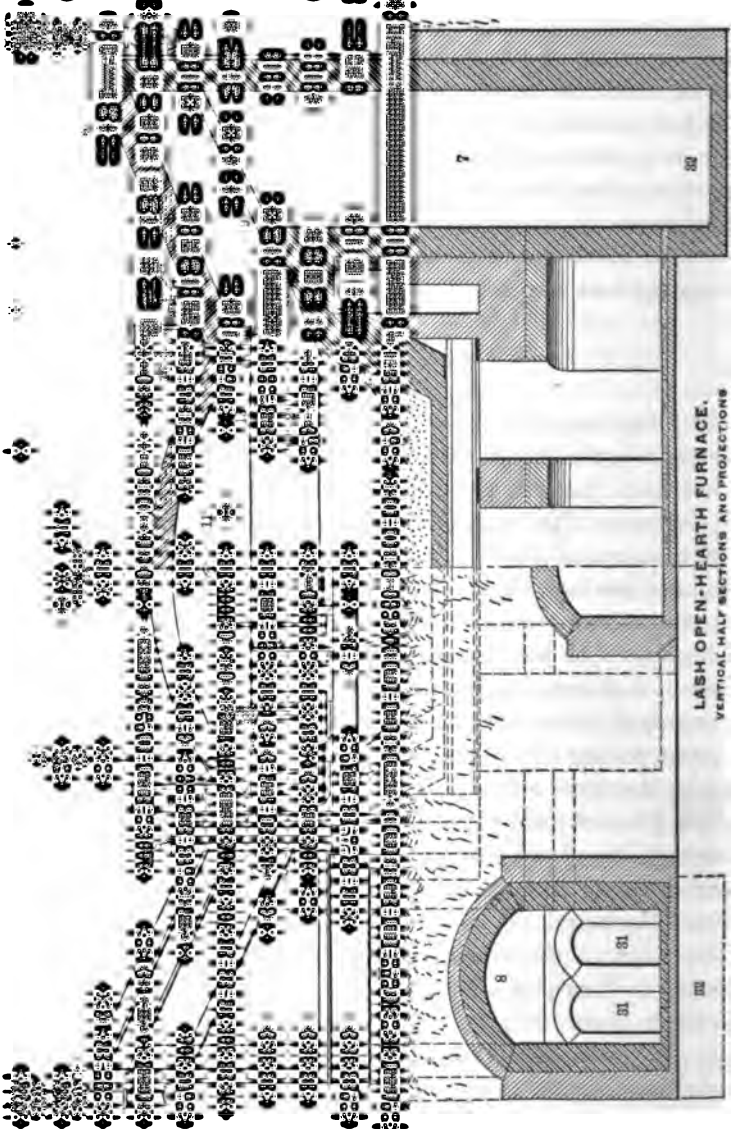
The latest improved Lash open-hearth furnace-plant, the writer believes to be the best yet devised. It is peculiarly adapted to the use of natural gas, and there are, at the present writing, 12 furnaces now erected in Pittsburgh on the Lash system, four of 40 tons, five of 30 tons, one of 20 tons and two of 15 tons, and there are four 15-ton furnace buildings. (See Figs. 1, 2, 3, 4, 5, 6 and 7, in which the parts are numbered for reference.)

The hearth of the furnace (1) is made circular or, preferably, elliptical, with major axis of 18 feet of hearth inside the linings and minor axis of 15 feet for a 15-ton furnace. This hearth is supported on beams, resting on suitable walls or piers, of such a height that the charging-doors are accessible from the ground-level. The retaining-shell of $\frac{1}{2}''$ thick steel plates extends from the top of the side-walls down to a broad connecting band (4) a short distance below the charging-doors; from this, the shell slopes inward to the bottom-plates of the hearth.

The lining of the hearth conforms to the shape of the shell, and since the bottom-plates and supporting beams prevent any vertical movement downward, the conical shape of the outer walls of the lower portion of the melting-chamber prevents almost all of the outward movement due to the expansion of the lining, thus preventing the rupture of the lower side-walls of the furnace.

The single flues (9) in natural-gas-furnaces at either end of the melting-chamber, are five feet wide and are simply large passages

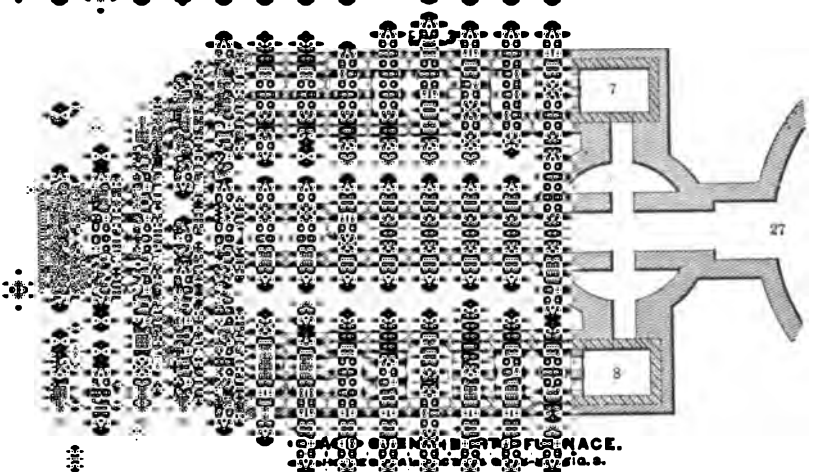
pitch of about four inches to
ide downward upon the metal.
r the arched roofs of the melt-



, a water-bosh, Fig. 7, made of
form of a keystone in the arch

and that over the
the inclined sides of
ouble skew-back or
bottom and five feet
by transverse bars,
ense heat of, perhaps,

he melting-chamber,
een the under-side of
n service for several
e furnace, which has
d from the surface of



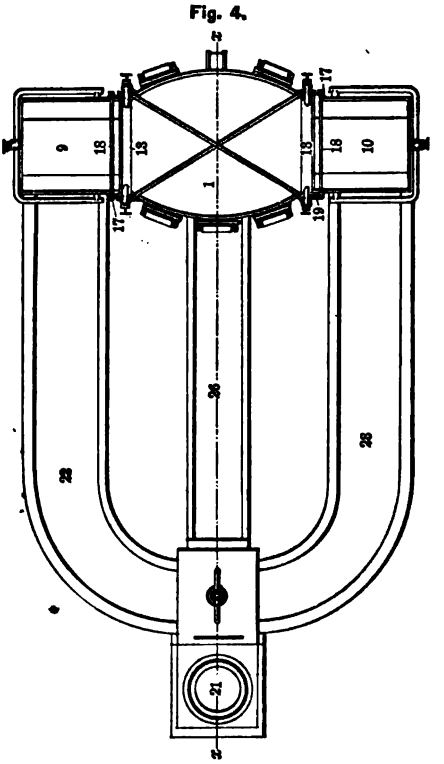
ACID OPEN-HEARTH FURNACE.

ded with 1-inch pipes
leading out the waste

The side-walls of the flues are held in place by suitable plates, angles and backstays (16) (17) (19), and the outer walls of the furnace are also stayed from opposite sides by means of plates reaching upward from the surrounding bands of the melting-chamber and diagonal rods (18).

Natural gas is led into the sloping flues by wrought-iron pipes (10–17) which enter the brick-work near the "four corners" of the

Fig. 4.



LASH OPEN-HEARTH FURNACE.
PLAN.

melting-chamber. The gas, being much lighter than the air, mixes with it in its downward rush into the furnace.

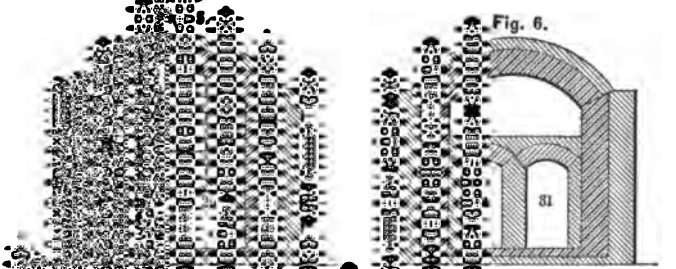
The stack (21) is placed in such a manner that the flues leading from each end of the hearth (22–23), which have checker-work in them, alternately act as regenerators to preheat the air before it enters the furnace. The lower end of the stack is connected by a short flue (24) with a four-way chamber (25), to which the flues (22–

to which the air-duct
m the ladle-pit (27),
way not only heating
earth and preventing

ng from the central
he furnace, is placed
), so that the bottoms
onger draught; but
r to ascend, there is
the furnace by this

ith a series of double
side-walls and tops

Fig. 6.



ON V-V. FIG. 5.

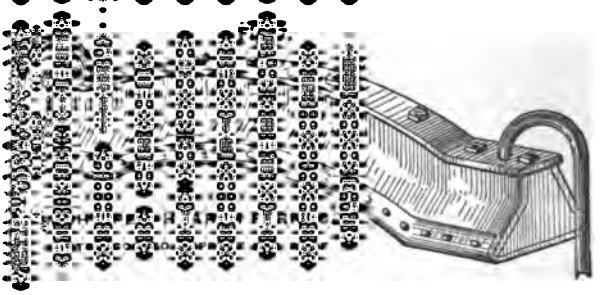
surfaces. The uptakes
ance of the chimney-
the reception of any
rawn over from the
from going forward
(31) in the chimney-
in the brick-work, so
e cinder-pockets may

(26) leading into it
valve (33) alternately
d of the furnace with
5), in this way revers-
nciple. This three-
by a stream of water

rping or burning out of the
direct loss of fuel by leakage

(34) is at about the ground-
ough an inclined spout (35)
(27). The ladle, after being
orted to an auxiliary semi-
lower than the ground-level
e front of the furnace is pro-
t they cover the entire pit-
furnace melting-chamber, so
urnaces, large pieces may be

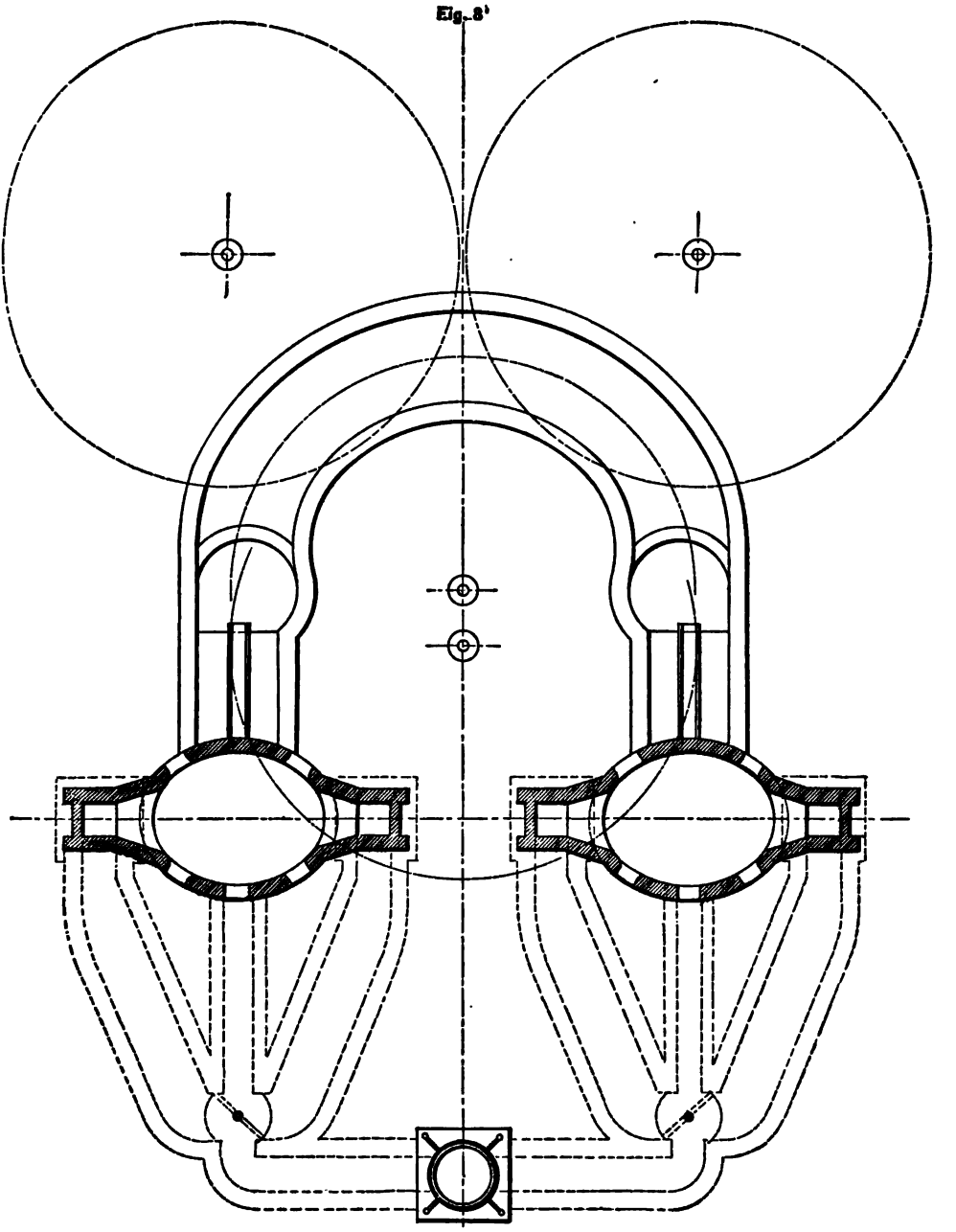
naces are:
important operations around
ree doors on the back-side of
or tapping-side being all ac-



e furnace. A record of 500
stock each, shows that these
utes per charge, 12 men, or
charging from all five doors,
open by levers in the pulpit

accessible from the ground-
rms and stages.
ranged as to be entirely isolated
except where the flues enter
necessity of thick brick walls
sides; the masonry being of
equal expansion and bulging

Fig. 8.



PLAN OF OPEN-HEARTH, TWO-FURNACE PLANT.

4. The chimney-flues, regenerators, three-way valve and ladle-pit are all on one level, about ten feet below the ground-level and easy of access for repairs.

5. The free access of air all around the flues and furnace-chamber prevents their being unduly heated at any time.

6. The rapidity with which repairs have been made upon Lash furnaces is one of the most important of their advantages. In two weeks' time, from heat to heat, a 30-ton furnace was repaired, the bottom taken out and the brick-work, from the ground-level up, put in entirely new, with seven masons working on day-turn, without any night-work except in tearing out the brick-work, and, in the five days, making bottom and reheating. Thirty-ton furnaces have been shut down and cooled off after the heat on Saturday, the furnace allowed to cool on Sunday, the roof and side-walls torn out on Monday, the brick relaid on Tuesday and Tuesday night, gas turned on Wednesday morning, and the furnace charged, with good results, by Thursday noon. That is a rebuilding of the lining of the entire melting-chamber, from the hearth up, and of the side-walls of the flues leading to the down-takes, with a loss of time of only five days from heat to heat. The cost for repairs per ton of steel ingots produced has been reduced to from 13 cents to 20 cents in the large forty-ton Lash open-hearth furnaces used in the Pittsburgh District.

A Lash open-hearth plant is now being built by the Wetherell Bros. Steel Casting Company, at Thurlow, Pa., for the use of producer-gas, with modifications necessary for the double sets of flues and checker-work, to preheat the gaseous fuel as well as the air, and double sets of ports, the air over the gas, for their entrance into the furnace. There seems to be no reason why the Lash system of furnaces, suitably modified, will not be equally advantageous for producer-gas as for natural gas.

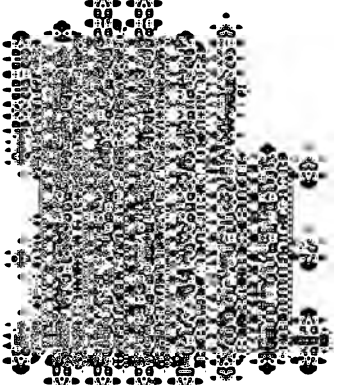
The Batho Furnace.

The Batho furnace (Figs. 9, 10, 11, 12) is one of the best forms of open-hearth furnaces yet devised. It was recommended and its advantages were described by Messrs. Wailes and Dick, before the British Iron and Steel Institute in 1887.

It is unlike the old-style open-hearth furnaces, which are put into

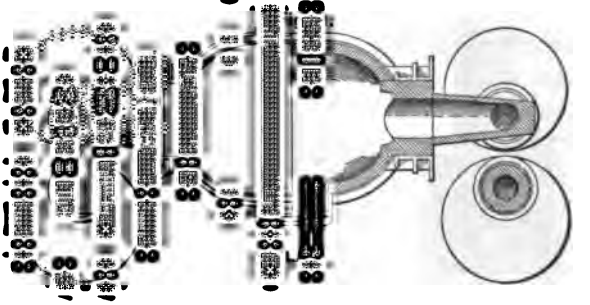
atus necessary to the
sists of five separate
fire-brick, which form
the melting-chamber.

g-chamber overhead
st horizontally, which
g-vessel is lined with
a brick enclosed in a
s suspended by rods
w inches; these rods



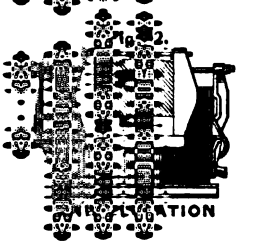
d girders, so that the
By this means either
more readily repaired;
could not be put in
the roof does not rest

rably to the endurance of both.
the melting-chamber, and the
he roof directly over the gas-
steep pitch into the furnace of



s arrangement guides the flame
es away with much of the sharp
of, which thus has to stand the
he basic lining is separated from
ral material in the form of car-

FIG. 2.



ATION

er 18 inches of the lining-walls
brick. The Batho furnace is
account of the facility of getting
round regenerator-cases have
ding from top to bottom of the
movable, rendering them very
ring. It is claimed that repairs
es very rapidly, records having
t 10 A.M. Saturday, the furnace
norning, when the regenerators
linings put into the furnace-
Sunday evening, and the furnace

prepared to charge for another 15-ton heat on Monday afternoon at 4 P.M. It is also claimed that the furnace heats up very rapidly, a cold 15-ton furnace being heated up ready to charge inside of twenty-four hours.

Acid Refractories.

In the production of acid refractories for open-hearth furnaces there have been few improvements made. Silica brick made of pure ground quartz, with 3 per cent. or a little less of well-calcined ferruginous lime, mixed to a milk of lime for a bond, and moulded under hydraulic pressures, which should be at least 10 tons to the square inch, are made by two or three concerns, and when well made have given good results. Considerable trouble, however, has so far been experienced with them, and hence they have not achieved the reputation that they ought to have. Often they have not been properly mixed. It requires careful and skilful manipulation to mix 3 per cent. of lime with 97 per cent of ground quartz in large quantities; and wherever the lime is in much greater proportion it acts as a flux and largely increases the fusibility of the mixture. Another fault has been that the lime used has not been properly calcined; it should be especially prepared for the purpose and calcined at a white heat to drive off all carbonic acid. Lime, as ordinarily sold in the markets, often contains much carbonic acid which by its disengagement, cracks and disintegrates the bricks when they are subjected to high heats. The bricks often have not had sufficient pressure in the moulds and have not been burned at a high enough heat. With the slight bond given by the small quantity of lime permissible to be used, a very high heat and a very heavy pressure in moulding are required, to make the bricks strong and able to stand the brick-axe well in cutting, and the large expansive action of silica in heating up. Properly treated, a silica brick can be, and has been, made in this country having a composition of about

	Per cent.
Silica,	97.50
Lime,	2.00
Oxides of iron and alumina,	0.50

which is fully as refractory as any of the best English or German silica bricks. By reason of its more even texture and much greater

strength, it can be much more readily cut and handled than the foreign silica brick.

In fire-clays and clay fire-bricks, although numerous brands are now on the market, but little improvement has been made in their quality over the bricks which were made when open-hearth furnaces were first introduced into this country.

Cost of Plant for a Given Output.

The developments in the open-hearth process have made it practicable to-day, in any desirable location, to build a 15-ton open-hearth furnace-plant complete, with an iron building 80 × 120 feet, equipped with suitable gas-producers, necessary cranes, hydraulic machinery, and all ready to run, for from $28,000 to $30,000, and with an additional cost of say $11,000 per furnace, three more furnaces can be built. For about $75,000 an open-hearth plant, capable of producing at least 180 tons of ingots per day, or an annual capacity of, say, 45,000 tons, can be built. This plant can be equipped with a universal blooming-mill at a value of $50,000; and, say a 23-inch, 16- and 10-inch mills, with iron building, at a cost of about $75,000 more, so that for $200,000 a complete steel-plant can be built.

The fact that an open-hearth plant complete can be built to turn out 45 or 50 tons of ingots per day for an outlay of $30,000, will probably, with the advantages of the basic hearth, allowing a much wider range of quality of stock being used, induce numerous small plants to be put up, especially in the Southern States, to produce steel, a state of affairs that will likely be felt by the comparatively few large concerns who now are nearly the sole producers of steel in this country.

II. Improvements in Process.

The Carbon Iron Co.'s Process.

The Carbon Iron Co. of Pittsburgh, has developed a new reducing process for the manufacture of wrought-iron direct from the ore, which seems to be, with the exception of the Russian Husgafvel process, about the only one which has been successful. The chief difficulty with all such processes has been, not to reduce the iron and to make wrought-iron from the ore direct, but to do this

without heavy loss. The Carbon Iron Co. uses as a reducing-agent (which also protects the reduced iron from reoxidation) a peculiar graphite from Cranston, near Providence, R. I. It resembles both graphite and anthracite coal, and has about the following composition:

	Per cent.
Graphitic carbon,	78.00
Volatile matter,	2 60
Silica,	15.06
Phosphorus,	0.045

The writer well remembers making prospective developments of this material in 1874, and reporting to his clients that it was for use as fuel, as compared with anthracite coal, a failure. The percentage of ash was not so great as to prevent its use near the mines, but it was found that it would burn only with extreme difficulty. In the Carbon Iron Co.'s process, this material is mixed with the iron-ore, both ground so that the particles will pass through a sieve of 16 meshes to the inch, and mixed with water so that the mass will mould in the hand easily, and in the proportion of weight of 2240 pounds of dry ore with 550 pounds of the graphite. This mixture is placed upon a large flat hearth of about 5 × 22 feet, with the stack in the center of the furnace. The furnace has for a bottom simply a layer of 8 inches of graphite upon the brick-work. The charge of mixed ore and graphite is spread over the furnace-bottom to the depth of about 4 inches, the doors of the furnace are well luted against ingress of air, and a reducing-flame of natural gas is allowed to play over the charge from jets at both ends of the furnace. In 20 minutes' time the mass has sunk to one-half its thickness upon the bed of graphite, and blue flames of carbonic oxide gas are seen coming up all over it, the temperature of the furnace being somewhere in the neighborhood of 1000° F. At the expiration of an hour, the charge has sunk to one-third of its former thickness, and beads of metallic iron are seen all over the surface. The heat is still kept with the reducing-flame upon the furnace for another half-hour, when the workmen, with ordinary puddlers' tools, commence the operation of balling-up. The reduced iron rolls up like snowballs in the furnace, and is easily made up in lots of 100 to 150 pounds, when each ball is rolled to one side and other balls are made up. Each furnace will produce, using Minnesota "Y" ore with 62 per cent. metallic iron, from about 12 to 15 of these balls, according to the size in which they

are made up, weighing from 10 to 12 per cent. more than the weight of the metallic iron charged in the ore. The heat at no time during the operation has been at all high; the furnace-hearth only getting to an orange-colored heat of probably 1500° F. towards the last of the operation, before the balling-up commences; and this orange-colored heat is more the reflection of the reduced ore on the melted slag than it is the intense heat of the brick-work and lining of the furnace. At no time in the production of the sponge for open-hearth stock is the heat raised to a sufficient degree to melt cast-iron; a welding-heat is, however, required if the balls are to be rolled. The stack is kept well dampered down during the entire work, as a reducing-flame is required for the success of the operation. This reducing-flame is easily obtained in the furnaces as used by the Carbon Iron Co., with an $1\frac{1}{2}''$ natural gas-pipe leading to a fire-box at each end of the furnace. The natural gas, being under a pressure sometimes as low as 4 ounces, the reducing-furnaces have several times run successfully when the pressure has been so low that the puddling furnaces in the adjacent mills have been obliged to shut down. All the balls are made up, in the ordinary practice, before any of them are withdrawn from the furnace. After all the balls are ready, which usually takes about one half-hour's time, they are ready to be withdrawn, squeezed in an ordinary rotary squeezer, and rolled into muck-bar or blooms, or taken to a hammer and drawn out into blooms, or, preferably, taken hot to the bath of an open-hearth furnace. The time taken for the entire operation is about two hours and a half, if the balls are to be used in an open-hearth furnace, or three hours if taken to the rolls. For open-hearth work each furnace will deliver eight heats per twenty-four hours of sponge-balls of a weight of 1550 pounds of balls per heat.

The principle of the process is that the peculiar graphite used as a reducing agent does not burn out rapidly, but adheres to the iron in considerable quantities in the sponge, protecting it and allowing the balls to become compact and solid, so that when they are taken from the furnace they do not reoxidize very rapidly, but can be squeezed and hammered or rolled without a comparatively severe loss. The oxidation in carrying the sponge-balls from the reducing-furnaces to the open-hearth, although the distance be considerable, is almost nothing. In fact, the sponge-balls can be taken out of the furnace and allowed to cool off completely in the air with but very trifling oxidation—less than that upon an ordinary compact

bar of iron of the same weight, and heated under the same conditions.

As a stock for the manufacture of open-hearth steel, the sponge-balls produced by the Carbon Iron Co. are peculiarly adapted. It is well known that the best quality of open-hearth steel is made by using a pure pig-metal, diluting it with the best quality of new iron blooms. American open-hearth managers would almost universally say that the best quality of open-hearth steel is made by using a good quality of pig-iron diluted with charcoal blooms, made in the open Catalan forges of the New York bloomaries. The sponge-balls made by the Carbon Iron Co. give the same excellent results produced with charcoal blooms, and have the advantage that they can be charged hot from the reducing-furnace into the open-hearth bath, in which they melt down as fast as they are put in, which is not only a great economy in fuel and labor of charging, but also in the rapidity in which the open-hearth operation is conducted, giving a saving of fully 50 per cent. in the time of making a heat and producing at the same time a better quality of steel. It is reckoned that heats of 15 tons of open-hearth steel can be made in four hours' time with the aid of a cupola to melt the pig-metal, and charging the Carbon Iron Co. blooms hot, while it takes eight or nine hours at the least with ordinary stock in open-hearth furnaces. In the practice which the Carbon Iron Co. is preparing to use, $\frac{1}{3}$ of the total weight of the open-hearth bath will be pig-metal first melted in a cupola and run into the open-hearth furnaces; and this will be diluted with $\frac{2}{3}$ of the total weight of Carbon Iron Co. sponge-blooms, which are added toward the last of the operation until the bath is run low enough to tap, the reducing operation being much better controlled by this process of dilution with hot sponge-balls than by the addition of iron-ore. In the pig- and scrap-process, when the gas-pressure is irregular, or when for any reason the heat is down during the operation, or bad holes are developed in the furnace bottom, or for any other reason it is desirable to tap the furnace without delay, the entire charge being in the melting-chamber, there is no other alternative than to go ahead and melt the entire mass down as well as possible. The nature of the Carbon Iron Co.'s process is such that the material under treatment is held, almost during the entire operation, in the melting-chamber, in such a state that at any moment it can be quickly prepared for tapping. This possibility will be highly appreciated by open-hearth managers.

The slag formed during the melting operation with the addition of carbon sponge-blooms is excessive, and foams up and boils freely on the top of the bath. At this stage of the operation it is easily removed in large quantities from the charge and allowed to flow out of the openings which have been specially prepared for this operation. In this way eight or ten buggy-loads of slag, weighing in all about 1500 pounds, can be removed from a 15-ton open-hearth heat. This slag does not contain over ten per cent. of metallic iron existing in it mostly as protoxide; very little metal is shotted through the slag at this stage. When this slag has been properly run off the bath, there is little trouble to be anticipated from the slag being in too large quantities on the bath in the after-operations, when it is necessary for the heat to permeate through the slag to the metal.

The percentage of loss of iron from ore to pig-iron made ordinarily in the blast-furnace runs about 5 per cent. It is safe to say that the loss of iron in the average anthracite or coke hot-blast furnace practice amounts to 7 per cent. Of course, this means the actual difference in metallic iron between ore and pig, and takes account of the fact that ordinary pig-iron has about 1.25 per cent. of sand upon its exterior and that its percentage of carbon and silicon are so high that it does not average over 95 per cent. of its weight in metallic iron. The loss of metallic iron in the puddling process in its product muck-bar, reckoning not only the iron in the pig added, but also in the iron-ore used as "fix," is from 12 to 15 per cent.; so that the loss by the ordinary blast-furnace and puddling process of manufacture of muck-bar is at least 15 per cent. of the metallic iron from ore to muck-bar. The writer believes the loss of metallic iron from the ore to average fully 20 per cent. in wrought-iron and fully 15 per cent. in the puddled blooms that are now largely used in the best open-hearth practice. The loss in the average open-hearth practice, using pig and scrap, is about 7 per cent.; using pig and scrap and puddled or charcoal blooms, at least 10 per cent.; and from ore to open-hearth steel the loss of metallic iron is from 17 to 23 per cent.

As will be shown later on, the loss of metallic iron averages in the product of over forty heats in a small experimental open-hearth furnace, about 20 per cent. from ore to steel ingot, and about 10 per cent. from ore to Carbon Iron Co. sponge-balls, which contain about 83 per cent. of metallic iron, on the average. Using 62 per cent. Minnesota "Y" ore, the average percentage by weight of the sponge-

balls of the iron-ore charged is 66.75 per cent. The weight of the sponge balls drawn is about 12 per cent. more than that of the metallic iron in the ore added. The loss in rolling into muck-bar is about 31½ per cent. of the iron contained in the ore. On account of the low heat at which the sponge-balls are squeezed and rolled, a considerable loss results from masses of the sponge falling off and not being squeezed and rolled into the solid muck-bar. This waste, however, is not of iron reconverted into oxide, but is largely metallic iron which has scrap-value of at least $22.00 per ton to remelt into steel in the open-hearth furnace, as it can be readily raked out and separated from the slag and oxide of iron. The percentage of the weight of the Minnesota "Y" ore charged of the Carbon Iron Co. sponge-balls is 66.75 per cent.; rolled into blooms is about 50 per cent., and into billets about 45 per cent., and into muck-bar of 3½" x ⅝" from 42 per cent. to 43 per cent. This means that for every hundred pounds of Minnesota "Y" ore of 62 per cent. metallic iron charged there is a yield of:

	Pounds.
Carbon Iron Co. sponge balls,	66.75
" " blooms 6 x 6 x 20",	50
" " billets 4 x 4 x 24",	45
" " muck bar 3½ x ⅞",	42.50

The following is a true copy of a week's run of three of the Carbon Iron Co. reducing-furnaces, the weights of which were under the writer's personal supervision, and he can vouch for their accuracy. The results this week were unusually good, the average loss being, as before stated, about 31.5 per cent. into muck-bar.

The regular charge for each furnace was 2240 pounds of Minnesota "Y" ore and 600 pounds of graphite, which gave the following weights of 3½ x ⅝" muck bar:

Furnace	No. 6. Pounds.	No. 7. Pounds.	No. 8. Pounds.
Monday, January 16th,	2965	2845	2910
Tuesday, January 17th,	2965	2810	2990
Wednesday, January 18th,	2815	3005	2955
Thursday, January 19th,	3015	2955	2875
Friday, January 20th,	2960	2040	2980
Saturday, January 21st,	1925	1865	1935
	16645	15520	16645

	No. 6.	No. 7.	No. 8.
Total number of heats made by each furnace into muck-bar,	17	16	17 = 50 heats.

Each of the above furnaces made three heats each day, except on Saturday, when they made but two.

	Pounds.
Total weight of iron-ore charged,	112,000
" " metallic iron in the ore,	69,440
" " muck-bar produced,	44,810
Loss,	20,630, or 29.71 per cent.

There is ordinarily very little slag taken from the reducing-furnaces after the sponge-balls are withdrawn for the open-hearth furnace, about 18 per cent. of foreign matter in the sponge-balls carrying out about all of the gangue of the ore, together with some graphite. On this account, the operation does not reduce the phosphorus to any considerable extent until the slag is removed from the open-hearth bath, which carries a large amount of it off. However, if the reducing-furnaces be heated hotter, as is necessary when the balls are to be rolled, then a considerable slag is formed, weighing from 300 to 400 pounds per furnace. This slag carries nearly the whole of the phosphorus, as very little, in fact almost none, is left with the iron, reduced at the low temperature of, say, 1500° F., employed in the reducing-furnace. The muck-bar from the Minnesota Y-ore carbon-blooms contains less than 0.015 per cent. phosphorus. Using Minnesota Y-ore, this slag, from the blooms when they are heated hot to roll, contains:

Iron,	from 30 to 50 per cent., as protoxide.
Silica,	from 24 to 30 "
Phosphorus,	from 0.10 to 0.15 "

The Carbon Iron Co. has a small experimental Lash furnace of about a ton capacity, which has been running more or less regularly for the past four months with mixtures of Carbon Iron Co. blooms. Below is a record of a heat made in the presence of Mr. Robt. Forsyth, Superintendent of the Union Steel Co., Chicago, which is a typical one of many that have been made with the same mixture.

	Pounds.	
Moorhead pig-iron,	870,	charged at 9.30 A.M.
16 Carbon Iron Co. sponge-balls,	2010,	" 10.45 A.M. to 11.45 A.M.
70 per cent. ferro-manganese,	24,	" 1.10 P.M.
Total charge,	2904,	tapped at 1.20 P.M.

The ore-mixture was charged into the reducing-furnace at 9.10 A.M., making four hours and ten minutes from ore and pig-metal to steel ingot. The steel was cast into a 14 x 14 ingot and contained:

	Per cent.
Carbon,	0.12
Phosphorus,	0.053
Manganese,	0.32

and had, when rolled and forged into an eye-bar 6″ x 1″ in section and pulled in a testing-machine:

	Pounds.	Per cent.
Tensile strength per square inch,	63,630	
Elastic limit per square inch,	42,500	
Elongation in ten feet,		17.58
Reduction of area,		35.1

The sponge-balls were made from reducing-furnaces Nos. 7 and 10. The entire product of furnace No. 10 in 12 sponge-balls of the following weight was put into the open-hearth furnace as follows:

Pounds.	Pounds.	Pounds.	Pounds.
90	150	130	95
155	130	135	155
85	185	110	120

Total, 1540 pounds.

From furnace No. 7 was charged 4 sponge-balls of the following weights: 115 pounds, 110 pounds, 125 pounds, 120 pounds, or 470 pounds additional, making a total of 2010 pounds in the open-hearth furnace.

The charge in each of the reducing-furnaces was the regular one, with 2240 pounds of Minnesota Y ore of 62 per cent. metallic iron and 550 pounds of graphite. The contained metallic iron in the ore added to each reducing-furnace was 1390 pounds. The loss in the reducing-furnace, as proved by the weight of steel produced from the sponge, was about 10 per cent., which, taken from the 1390 pounds of iron, would leave 1250 pounds of iron in the 1540 pounds of sponge; or the sponge contained about 81.23 per cent. of metallic iron, estimated in this way. It is very difficult to get by analysis the percentage of iron in the sponge, which is not at all homogeneous as to the amount of contained slag and graphite. The steel ingot weighed 2150 pounds, and the scrap 191 pounds,

yielding a waste of 563 pounds; or, in percentages of the total charge of 2904 pounds:

Ingots,	74.00 per cent.
Scrap,	6.58 "
Loss,	19.42 "

The small open-hearth furnace has proved by charges of regular pig-and-scrap mixtures to give a loss of about 11 per cent. The same mixtures used in Lash open-hearth furnaces has yielded about 7 per cent. loss, giving an excess of about 4 per cent. loss for this small experimental open-hearth furnace over that which would be obtained ordinarily with a large open-hearth. This excess of 4 per cent. can fairly be subtracted from the percentage of loss given above, 19.42 per cent., yielded by the small furnace, giving a total loss of between 15 and 16 per cent., which can be safely predicted to be the loss by the process in actual open-hearth practice in fifteen-ton furnaces. Of this loss, from 6 to 7 per cent. is the loss in ordinary open-hearth practice, and from 9 to 10 per cent. is in the reducing furnace. This loss of iron, of say 16 per cent., it must be borne in mind, is from ore and pig-metal to steel ingots, and is largely loss of iron from the ore, and not a loss of iron from a manufactured product. The cost per "unit," or per cent. of iron, in this loss is, at say $11.30 per ton for the iron in the ground ore, while the cost per unit for the loss by the pig and scrap process is at the direct values per ton of the stock used, at least $20 per ton.

The Minnesota Y-ore used in the above experimental charges contains:

Iron,	62.16 per cent.
Silica,	5.54 "
Phosphorus,	0.063 "

With a charge of	2000 pounds	Carbon sponge-balls, and
	1000 "	Castle pig-iron.
	3000 pounds,	total charge in an open-hearth furnace.
Estimating a loss of 17 per cent. in the open-hearth process from charge to steel,	510 pounds,	loss,
Will give a yield of,	2490 "	steel produced.
Reckoning the yield from the pig at,	930 "	or 7 per cent. loss on the 1000 pounds of pig.
We have a yield of,	1560 "	steel from the 2000 pounds of sponge.

In the reducing-furnaces, 2240 pounds of Y-ore yields 1560 pounds sponge; or, 1 pound of sponge takes 1.445 pounds of ore to produce it. According to this, the 2000 pounds of sponge in the above calculation would require 2890 pounds of ore to produce it. This quantity of ore, at 62 per cent. metallic iron, contains 1791.8 pounds of iron.

1791.8 pounds of iron in the ore, less
1560.0 " " in the soft steel produced from this sponge.

231.8 pounds of iron lost in the manufacture of the ore into steel from the sponge.

This is 12.9 per cent. of the 1791.8 pounds of iron in the ore.

The steel made by this company from sponge-blooms, in an open-hearth furnace, certainly has given the best results yet obtained for structural purposes, and has already attracted favorable comment. The *Engineering News and Railway Journal* has lately devoted over a page to some of the results obtained from it in the shape of angles, plates, and eye-bars. The rapidity with which it can be made, and its economy of materials, as compared with other processes making anything like the same grade of steel, indicate a successful future for it.

The Husgafvel Process.

The Husgafvel process for making iron direct from the ore for open-hearth furnace-stock, the writer will only briefly mention, since another paper on this subject will be read by Mr. Garrison.* Briefly, the Husgafvel process of making wrought-iron is a modification of the old *Stückofen* process, by putting on a movable hearth and rendering the working continuous. An iron reinforcing-band is also put around the tuyeres, and a hollow iron jacket around the upper walls of the furnace, through which the blast is carried, in this way heating it to 120 degrees. The heated blast insures the production of a good welding-heat in the hearth. The use of a blast-heating jacket keeps the walls of the furnace cool, and prevents their being burned out, or the lodging of partly reduced masses in the upper portion of the stack. The fuel used is charcoal, four parts charcoal to from six to ten parts of ore; the reduction of wrought-iron taking place at a very low heat, and when the bloom has reached the weight of about 150 pounds, the

* See page 334 of this volume.

hearth, which is on a car on a track, is removed, and a new hearth put in place for the next operation; the bloom being lifted out of the hearth and taken direct to the open-hearth furnace-bath. The loss of iron, from ore to iron in sponge, varies from 10 to 15 per cent., and from ore to steel is from sixteen to twenty-five per cent. This process has been in successful operation since 1875, and it is interesting to note the close similarity of the results as to waste, from ore to steel, obtained by this process and those obtained by the Carbon Iron Company.

The Pig-and-Ore Process.

The pig-and-ore process, without the addition of more than 20 per cent. scrap, is now very little used in this country; since it takes at least 12 hours, or more than 25 per cent. longer time to make a heat by this process, and the percentage of loss is, in the best practice, from twelve to fifteen per cent. of the contained iron in pig and ore added. In the pig-and-ore process, a pig low in silicon should be selected, and ore, rich in metallic iron, also low in silica. The African Mokta ore and the native Republic hematite ore are among the best used in this country. It usually takes from 15 to 20 per cent. of the weight of the pig, of ore of from 60 to 65 per cent. iron, to reduce the metal to carbon 0.15 per cent. The pig-and-ore process, when properly worked, certainly gives a very superior product; and as it is worked in England, especially, it has made remarkably good steel for structural purposes. Its great disadvantages here are that it requires many more hours of the work of many skilled furnace-men who are paid comparatively large wages, and that during this extra length of time the charge is in the furnace chamber at intense heat, which is very destructive to refractories.

Basic Open-Hearth Process.

The basic open-hearth lining necessary for the existence of a basic slag over the bath of molten metal must extend over the entire hearth and up the side-walls high enough to be at least above the slag-line. The brick for this lining are made of either dolomite or magnesite. The composition of the various grades of dolomite that have been used in making basic brick in Germany, France, and England, and that are proposed to be used in this country, is given in the following table:

	Germany.			France.			Austria.	Poland.		United States.		
	Hörde.	Ilsede.	Rothe Erde.	Varigey.	Dion.	Besèges.	Chozanow.	Dombrowa.	Scotch.	Virginia.	Pennsylvania.	
Silica SiO_2	2.02	1.35	0.60	4.10	0.10	3.80	0.80	2.00		15.40	3.10	15.30
Protoxide of iron FeO	—	0.26	—	—	—	—	—	—				
Sesquioxide of iron Fe_2O_3	2.30	2.05	1.16	1.70	4.00	4.00	0.90	3.20		5.88	1.50	5.40
Sesquioxide of aluminum Al_2O_3				3.00	1.50		4.10	1.30				
Lime CaO		30.12	32.45	28.30	31.40	28.00	29.00	31.00		27.12	48.20	27.29
Magnesia MgO		19.21	19.15	18.60	16.40	17.00	17.30	16.10		17.48	2.74	14.18
Moisture H_2O	—	1.99	—	—	—	—	—	—				
Carbonic acid CO_2		44.97	46.45	44.20	42.20	45.00	46.20	45.40		35.24	40.90	37.05
Carb'ate of lime $CaCO_3$	61.31										86.08	48.74
C'rb'ate of magnesia $MgCO_3$	34.42										5.76	29.78

Dolomite brick are much cheaper than magnesite brick, but they do not give as good results, for dolomite is very apt to be irregular in chemical composition, and bricks made from it do not stand the weather well; in the furnace they are not as durable as magnesite bricks and cannot be put in direct contact with silica or other acid bricks as magnesite bricks can. The composition of bricks made of dolomite is as follows:

	Germany.		France.	Austria	Pennsylvania.
	Hörde.	Ilsede.	Dion.	Chozanow.	
Silica SiO_2	5.95	4.86	2.30	3.10	7.40
Sesquioxides of iron and alumina	4.30	3.95	8.20	2.60	3.25
Lime	56.15	59.10	60 10	61.25	51.05
Magnesia	32.40	30.03	30 01	31.60	37.20

In the basic open-hearth practice the weight of calcined magnesia used for repairs to the lining and bottom of the hearth, and in producing the basic slag, is about two and one-half to three per cent. of the weight of the steel produced, or about fifty pounds per ton of steel. With the use of dolomite nearly three times as much is needed as with magnesite, or about 150 pounds per ton of steel, or about 8 per cent. Unfortunately, good magnesite is a somewhat rare mineral. In this country there are but few places where it can

be obtained, and at present the source of our supply is almost entirely from abroad. Magnesia (MgO) can be manufactured, but owing to the expert labor required to look after the chemical reactions, it is too expensive. The methods of manufacture are briefly: *

1st. By Scheibler's process, which consists of the action of molasses on an emulsion of calcined dolomite, which produces soluble saccharate of lime and insoluble magnesia, which is filtered off.

2d. Closson process,—the reaction in water of chloride of magnesium and calcined dolomite,—which produces soluble chloride of calcium and insoluble magnesia, which is filtered off.

3d. The evaporation of sea-water, the residue of which contains from fourteen to fifteen per cent. of sulphate and chloride of magnesia, which is treated with milk of lime, which produces soluble chlorides and sulphates of lime and leaves insoluble magnesia.

4th. Shrunk dolomite is put into an emulsion in water in a strong iron vessel and carbonic acid is forced in under pressure, which forms soluble bicarbonate of magnesia, the lime remaining insoluble as carbonate. The solution is filtered, the excess of carbonic acid driven off in the filtrate by boiling, and pure carbonate of magnesia precipitated and obtained by filtration.

Magnesia (MgO) obtained as a manufactured product, 95 per cent. to 98 per cent. pure, costs in this country now about $35 per ton. The following is the composition of the various native magnesites giving the best results, and of the commercial magnesia carbonate, as manufactured by one of the processes described above.

	Austria. Steiermark.	Greek Island of Euboea.	Styria. Mitterdorf.	Saxony. Frankenstein	U. S. California.	U. S. Manufactured.	2d Sample. Greek Island of Euboea.	U. S. From Rhode Island.
$MgCO_3$ carbonate of magnesia	95.50	96.00	92.20	93.00	94.69	98.60	94.40	97.46
$CaCO_3$ carbonate of lime	1.00	2.30	2.50	4.00	none	0.10	3.75	
SiO_2 silica	1.00	0.50	1.00	1.00	2.40	0.90	0.88	0.18
Fe_2O_3 and Al_2O_3 oxide of iron and alumina	2.00	1.20	4.30	2.00	0.38	0.40	0.37	0.35
MnO oxide of manganese, H_2O water	0.50							0.28

Brick made from some of the above magnesites have about the following composition:

* See Prof. Egleston's paper on "Basic Refractory Materials," *Trans.*, xiv., 458.

	Euboea Magnesite, Brick, Sample No. 1.	Euboea Magnesite, Brick, Sample No. 2.	Mitterdorf Magnesite, Brick, Sample No. 1.	Mitterdorf Magnesite, Brick, Sample No. 2.	Steiermark Magnesite, Brick.	U. S. Manufactured Magnesite, Brick.	U. S. Rhode Island Magnesite, Brick.
Magnesia MgO	88.90	87.60	81.23	82.10	86.20	91.10	90.40
Lime CaO	3.40	4.80	0.50	1.15	0.75	0.25	0.15
Silica SiO_2	3.90	2.60	8.63	7.65	2.95	2.85	2.15
Oxides of iron and alumina, Al_2O_3 and Fe_2O_3	2.40	2.80	8.40	8.20	5.70	1.70	4.80

Euboea magnesite is almost absolutely infusible; it is calcined in a cupola furnace satisfactorily. The Mitterdorf magnesite requires a reverberatory furnace, as it calcines to a very light dust, which mixes with the coal-dust and is lost. The magnesite from Steiermark probably makes the best magnesia bricks of all yet discovered. It has a peculiar structure, different from any of the others calcined. It has a specific gravity of only 3.6, and requires a very high degree of heat to dead-burn it, as it must be kept at a white heat for some time to completely expel the carbonic acid. When dead-burned it shrinks 25 per cent. in volume, and changes its physical structure entirely, will not slack or change in the atmosphere, has very little bond or sticking capacity. The dead-burned magnesite of Steiermark is solid and compact, and has been kept for years without absorbing carbonic acid. Brick made from dead-burned magnesite shrinks only 4 per per cent.

For the manufacture of brick the magnesite is calcined in two lots; the first, 90 per cent. of the whole, is calcined at a very high heat for a considerable time, and ground so that it will pass through a sieve with a mesh of one square millimeter. The second lot, to be used as a bond, to the amount of 10 per cent. of the whole, is not calcined at such a high heat, but is kept for quite a long time at a heat, such as not to frit the material or fuse the contained oxide of iron, but sufficient to drive off fully 80 per cent. of the carbonic acid. This lot is ground very fine, to an impalpable powder, and mixed with just enough hot water to moisten the whole lot after the 90 per cent. of dead-burned material is added. The whole mass is well stirred together and immediately pressed into bricks, as it sets quickly. The bricks are made abroad in a size of 11″ x 4½″ x 3″, or 270 x 130 x 65 millimeters, and are pressed by hydraulic pressure equal to about 150 tons applied on the 11″ x 4½″, or about 3 tons

per square inch; the pressure being applied by the stroke of a hydraulic ram. Bricks made of magnesite in this way need no foreign binding-material, which is a very decided advantage, as any additions of clay, lime, alkalies or the like render the mass fusible. Magnesia bricks weigh about 150 pounds per cubic foot. Pure calcined magnesite, when made into bricks correctly by the process described, has not been melted by any heat yet reached, and is not acted upon by any basic slags or slags rich in metallic oxides. These facts will mean much to metallurgists in the future; since these properties enable us to be almost independent of our refractory linings in the limits of temperature which we can produce. In the past it has been a question of refractories; in the future the upper limit of temperature may be a question merely of the dissociation of gases used as a fuel. Magnesia bricks, properly made, stand the weather well; they can be kept indefinitely, so that they may be procured in large quantities, and if reasonable care is taken in protecting them from water, there need be no fear of their decomposition. It is not a fair test to put a single brick upon the bridge-wall or bank of a furnace to determine its quality. To test the wearing quality of magnesite bricks, they should be actually built into the work at the most exposed parts of a furnace and their behavior noted there. Magnesia bricks are best laid in a mortar composed of very finely ground magnesia brick, together with some of the same bond-material as the bricks are made of, the magnesite calcined at a low heat and ground very fine, and mixed with just enough water to make the mass plastic. The bricks are also well laid in a grout of ground brick, mixed with enough tar to make it plastic, the mixture being kept in iron pans kept hot with fire underneath, and the trowels used to lay the mass being kept hot to prevent sticking. The advantage of this way of laying the brickwork is that the furnace is ready for heating up much more rapidly than when water is used to make the cement plastic. Where the acid and basic bricks meet, no mortar of any kind is needed; the bricks are simply placed brick to brick. With dolomite brick, it is necessary to use at least one-half inch in thickness of some neutral parting material. Chrome-ore and carbon brick have been used, the preference being for the carbon brick, as the chrome ore softens and allows the brick work to settle, especially when steel gets into contact with the chrome-ore joint. Snelus, Wailes, Dick, and quite a number of English metallurgists are on record as preferring the

carbon brick to the chrome ore for the neutral joint. The hearth made of magnesia can be set to any desired shape, the contained 3 per cent. of silica, alumina and iron acting as the necessary flux. To do this, however, takes considerable time and considerable skill, as the magnesite is set in layers of not over 3 millimeters thick, each layer having to be set by heat, taking, on the average, thirty minutes before another is added; and as there are from 100 to 300 of these layers, it takes over a week's time to make a magnesia bottom. If one layer is not properly set, it is liable to cause trouble by rising and throwing off all the layers above it. However, with a magnesite bottom once properly set, it is very durable, requiring but little repairs. The thickness of the magnesite hearths varies from 300 to 400 millimeters, on the avérage, in the German furnace. Fine iron filings are added on curved places to make the magnesia set more strongly and rapidly. The roofs of the furnaces are held in a beveled metallic ring acting as a skew-back and supported from walls outside the basic lining, so that the weight of the roof is kept entirely off from the interior heated walls of the melting-chamber. In the most approved smaller furnaces these roofs are suspended from an overhead railway, so that by the aid of turn-buckles the roof may be entirely lifted off of its connection with the walls of the furnace and carried horizontally away from the furnace-chamber, in this way greatly facilitating repairs to the furnace-chambers. The top ten to fifteen inches of the side-walls of the furnaces are of silica brick, it being only necessary to have the joint between the basic and acid brick well above where the action of the metal and slag can attack it. The tap-hole in the basic hearth is built of basic brick arched and forming a tunnel up into the furnace-bottom. It is stopped by placing a stopper-iron in the ordinary way through the back furnace-door against the inside of the aperture. From the spout a mixture of perfectly dead burned magnesite and tar is placed against this stopper for a thickness of about 5 inches, and hammered with wooden sledges, to make it compact. From the outside the hole is filled up flush with the outer walls with a mixture of sand and tar, which can be easily cut out when ready to tap. The total length of the tap-hole is usually about 12 or 15 inches to the outside spout, which is, as usual, lined with acid brick, well grouted with clay. The durability of a magnesite hearth, if it is well taken care of, is well-nigh everlasting, even with pig- and ore-charges, which does not cut basic bottoms as badly as they do acid ones. In the pig-

and ore-process it is claimed that over 30 per cent. more ore can be used upon a magnesite bottom than upon a silica hearth. One of the points requiring extreme care is to cut out of the magnesia bottoms any drippings from the silica roof, as when the next heat is made, the steel cuts into the hearth and makes bad holes wherever these drippings are. It will probably be found advisable to make basic furnaces, roof and all, of magnesite brick, on account of their extreme infusibility. In Westphalia, the price per ton of magnesia brick is about $35.00, and of burned magnesia about $13.00, per ton. The writer has bought magnesia material, delivered in New York, at prices per ton as given below:

For magnesia bricks,	$38.00
For burnt magnesite,	16.00
For crude magnesite,	9.00

The ton calculated at 2240 pounds. The duty was 20 per cent. on magnesia bricks, 10 per cent. on burnt magnesite, and crude magnesite, free.

The basic open-hearth charge needs a pig-iron low in silicon, as the silica produced by it has to be neutralized, and, when in excess, throws up a thick heavy slag and retards the operation. The percentage of sulphur also should be low in all of the stock, as the basic slag does not eliminate sulphur as it does phosphorus. Some of the sulphur is expelled, but by no means all of it. When the sulphur is very high, about 30 or 40 per cent. of it is usually removed; but when the sulphur is below 0.15 per cent. it goes very slowly and is not usually run out below 0.10 per cent. In the basic Bessemer process, the best results are obtained with phosphorus up to 2 per cent. and over, metal rarely obtained from the ores in this country; but in the basic open-hearth any contained amount of phosphorus can be advantageously worked—in fact, the lower the percentage of phosphorus, the better the results that are obtained. It is a mistake to suppose that high phosphorus is required in the basic open-hearth practice. On the contrary, as in the acid open-hearth process, the better the stock used, the better the results that are obtained, although a very good soft steel product can be made with about 0.05 per cent. phosphorus, using the pig-iron made from our native ores with phosphorus running as high as, say, 0.30 to 0.35 per cent. The average open-hearth mixtures that could be selected in this country without reference to the contained phosphorus of all ore, native pig-

irons and scrap, would be found to contain, were they melted on an acid hearth, about 0.15 per cent. to 0.18 per cent. phosphorus, Such mixtures would yield, in a basic open-hearth furnace, good steel containing carbon at 0.15 per cent. to 0.20 per cent., about 0.03 per cent. phosphorus; and, with dead soft steel, as little as 0.01 per cent. phosphorus. With the basic slag the phosphorus which was in the iron forms tetrabasic phosphate of lime; the product of the reaction of the phosphorus and silica existing in the slag is a compound of about the following composition: $4Ca_3P_2O_8 + Ca_3SiO_5$, according to Harbord in his paper on the subject before the British Iron and Steel Institute, in 1887. Harbord also claims, in his paper, that alumina, Al_2O_3, in the basic slag of an open-hearth furnace acts as an acid and retards the basic reactions.

In making steel in a basic open-hearth furnace, a quantity of calcined lime (much better in its action than limestone) and of oxide of iron, either in the form of hammer- or roll-scale, or some calcareous ore which has very little silica in it, is strewn all over the furnace hearth; upon this the charge of pig and scrap or blooms is placed. In the pig- and ore-process the reaction is hastened and a great advantage is obtained in that a quantity of the ore can be added first, which melts first and forms a bath of oxide of iron, into which the metal flows as it melts down, and is actually puddled and rapidly reduced in carbon as the further melting operation is going on. On an acid hearth the ore added at first would rapidly cut out the bottom, and as a consequence it cannot be added until the melting is almost complete; and during the long time while the melting is going on, the portion already melted remains comparatively inert, so far as the reduction is concerned. During the continuance of the melting operation, an occasional quantity of lime and oxide of iron is added to keep the proper bulk and action in the slag. When the metal gets down to about 0.18 per cent. carbon, the reaction often gets slow and sluggish; in such a case, additional heat is put on the metal, and one or two pigs of metal, high in carbon, are added, which will start a more rapid boiling and expedite the removal of the carbon and phosphorus. The average time of making heats in the basic open-hearth furnace of about 15 tons capacity, charged with pig and scrap, is from seven to nine hours, probably a little more time than on an acid hearth, as the basic slag, when the metal gets low in carbon, usually is a little more thick and sluggish.

Basic open-hearth furnaces have been in successful use in England since 1885, and in Continental Europe still longer. In this country,

however, they have been run in a desultory way only. The Otis Iron and Steel Co., of Cleveland, O., is the only concern which has made any basic open-hearth steel on a large scale. The writer has done little work at a small experimental furnace; and at present, Carnegie, Phipps & Co., Ltd., of Pittsburgh, are building a basic open-hearth furnace. The writer, however, ventures to prophesy that in the future, just as the neutral lining for puddling furnaces has superseded the old sand bottoms, so will the basic open-hearth supersede the acid linings, especially if a good grade of magnesite at reasonable rates can be secured, for the reasons:

1st. That a steel can be obtained very low in phosphorus, even with the use of material containing important proportions of phosphorus.

2d. That the reason given by G. J. Snelus, in England, is undoubtedly true, that silica has a predisposing action facilitating the oxidation of iron, and that it makes the iron "tender" to the action of oxygen, while lime and magnesia, as bases, have the very opposite effect, retarding the oxidation of iron; the chemical explanation being that in the basic case, lime forms the base and peroxide of iron the acid, forming ferrate of lime, while in the acid case, oxide of iron forms the base and silica is the vehicle which seizes hold of it, forming silicate of iron. For this reason, a better yield is produced by the basic process and a better quality of product; for getting rid of oxide of iron is one of the necessities of good steel.

3d. The best basic refractories are much more infusible and much less acted upon by the slag and oxides of iron, and are much more durable than the acid materials, making the life of furnaces much longer and the out-put per ton of refractories consumed much larger.

The basic open-hearth, as a dephosphorizing process, is especially adapted for making dead soft steel where, with a magnesia bottom and a basic slag, 98 per cent. of the contained phosphorus in the stock, can be eliminated. For the manufacture of very soft grades of steel, it will very soon prove its superiority over the Clapp-Griffith modified Bessemer process, with its large percentage of loss. The metal has to be reduced in the basic furnace to .10 per cent. carbon, in order to reduce the phosphorus below 0.05 per cent. It is not likely that the basic process can be advantageously used for making pure steel from high phosphorus stock for the best grades of open-hearth work; at least, not for the present. Engineers have a very general and well grounded belief, that safe and homogeneous structural steel

should contain less than 0.03 per cent. phosphorus; and basic-steel makers who commence with a stock high in phosphorus will have great difficulty in getting engineers to accept the steel made from it without many repeated tests and analyses of every heat for phosphorus. In the past, *Lloyd's Register* and the Board of Trade of Great Britain have specified against the use of basic steel, and are now only accepting it in a very guarded way and after rigid inspection; and when it begins to be used in this country, as it surely will in the course of the next year, probably the basic open-hearth will, at least for a time, be tabooed here as a dephosphorization process for the manufacture of structural steel, except for rivets and for dead soft fire-box and boiler-plate steel. By this, it is not meant that the basic open-hearth process *per se* will be objected to, but that the practice of using stock high in phosphorus, trusting the action of the basic slag to produce the best grades of open-hearth steel from it without regard to the material used in its manufacture, will be objected to. In the natural course of events, manufacturers will attempt this and engineers will object to it, and will probably have experiences which will justify their objections.

In Sheffield, England, it has been urged that basic open-hearth steel with phosphorus at 0.017 per cent., or less, can advantageously replace the pure Swedish iron bar which enters largely into the mixture for the best grades of crucible steel. Undoubtedly the suggestion is a good one, and the material thus referred to can be usefully employed in the crucible-steel process; but the writer's experience in kindred lines is to the effect that, in making the best grade of steel in either the crucible or open-hearth process, new iron made by one of the direct processes gives the best results.

Undoubtedly, the basic open-hearth process, using good stock to start with, will yield the purest metal in metallic iron of all commercial methods of manufacture. Metal can be regularly made in the basic open-hearth, using good stock, with 99.75 per cent. metallic iron and with 0.18 per cent. of the residue manganese; but such pure iron has but limited uses, at least at present. It has a tensile strength of about 50,000 pounds per square inch, or about that of good wrought-iron, in direct tension. It, however, has the advantage of giving, on the average, fully 80 per cent. of its strength in direct tension if tested transversely, while wrought-iron, except in wide plates, gives very much less strength if tested transversely. The highest results in ductility have been obtained with this soft steel with about the following tensile tests:

Ultimate strength per square inch,	50,000 pounds.
Elastic limit per square inch,	27,000 pounds.
Elongation in 8 inches,	35 per cent.
Reduction of area,	75 per cent.

These are the maximum ductility results that have come to the writer's notice, and his experience is that when the tensile strength has been reduced much lower than 50,000 pounds per square inch, the ductility has not been increased, but rather decreased, and that the lower tensile strength is due to the metal not having received proper work, or to oxidation or some other fault with the steel. Mr. Wailes states before the British Iron and Steel Institute that, in his experience, the very soft and pure basic open-hearth steel does not need the excessive mechanical compression that ordinary steel does, and consequently smaller and less expensive blooming-mills will be required to break it down. As the steel has remarkably good welding-capacity, this, very likely, may be true.

The writer has lately been squarely controverted in a statement that soft steel may be easily and safely welded, in a paper before the American Society of Civil Engineers, by Mr. William Metcalf, and he therefore wishes to be very conservative and cautious in the face of the judgment and experience of such an eminent engineer. However, he feels it but just to the facts in the case to say that soft basic open-hearth steel sticks together with a great deal of tenacity, when properly heated and hammered together at this comparatively high heat, and the forging kept up until the metal has been considerably reduced in temperature.

It has been claimed before the British Iron and Steel Institute, that soft basic steel can be easily fagotted, bloomed and welded into scrap bars which are sound and possess all the qualities of the original steel; and President Adamson, in his annual address, in May, 1887, said that he personally had a considerable record of the free-welding properties of mild steel, and that at his works they were doing it successfully every day, and that he had found that steel to weld satisfactorily must answer the following chemical requirements: "That the carbon must be low, the manganese four times as much as the carbon, and that the silicon, phosphorus and sulphur combined must not amount to more than one-tenth of one per cent."

NOTES ON THE TOPOGRAPHY AND GEOLOGY OF THE CERRO DE PASCO, PERU.

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(Boston Meeting, February, 1888.)

The Mining Belt of Peru.

The great mining region of Peru is a mountainous belt of country, running nearly the whole length of the republic, and comprising the two grand ranges of the Andes with the elevated table-lands between them.

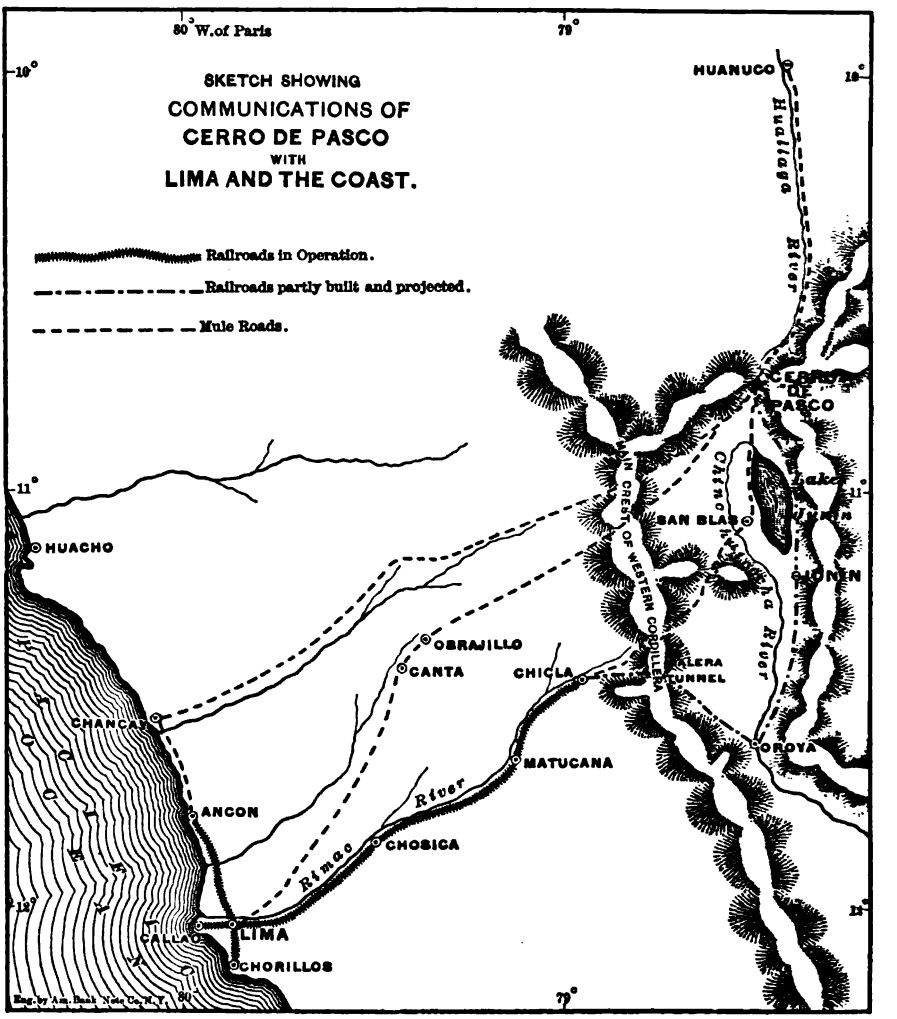
On the east of this belt are the extensive plains and fertile valleys of the Amazon and its tributaries. On the west is a narrow strip of coast 20 to 50 miles wide, for the most part a sandy desert, but producing abundant crops where irrigated; and here are found petroleum (Department of Piura), salt (at Huacho and elsewhere), nitrate of soda in enormous amounts at the south, silver in a few localities (Province of Tarapacá), copper and other mineral products.

The Western Cordillera, running nearly parallel with the shoreline, rises like a wall on the eastern side of the coast-belt, with passes from 15,000 to 18,000 feet high and peaks attaining 18,000 to 20,000 feet. Further east, at a varying distance, is the Eastern Cordillera, composed of broken mountain ranges which, considered as one group, have a general parallelism with the western chain, but individually vary greatly in direction, sometimes running nearly east and west, in places projecting out into the Amazonian lands, or here and there curving around to unite with the western ridge, and with this enclosing immense inter-alpine plains. These general features are indicated in Fig. 1.

The surface of these plains is uneven and traversed by lower ranges of hills which surround large lakes, or rolling pampas, or fertile valleys; and through many of these last run rivers of considerable size. The whole country has a high altitude (averaging up to 15,000 or 16,000 feet), and slopes gradually north and east towards the Amazon, into which drain all its waters. Its bounding mountain chains are scored on all sides by narrow, picturesque and pre-

cipitous ravines, often thousands of feet deep, where streams flow down and where flowers and fruits grow in abundance, in immediate proximity and wonderful contrast with the barren and dreary places above them. As yet only one railroad (the Arequipa-Puno road in

FIG. 1.



the south) traverses its realms. For the rest, excepting the unfinished Oroya road, mule-trails form the only means of communication.

In all parts of this region are deposits of valuable minerals. Gold and silver have been mined from the earliest periods; the

quicksilver-deposits of Huancavelica were once famed throughout the world, although now practically abandoned; lead- and copper-ores have been worked to some extent; salt and coal have been discovered at many points; and iron and other useful metals are said to exist. Notwithstanding all the drawbacks arising from want of roads, of proper methods and appliances of mining, of skilled labor, of capable management and of capital, this region has produced, according to Humboldt, an average annual yield of $5,300,000 in gold and silver.

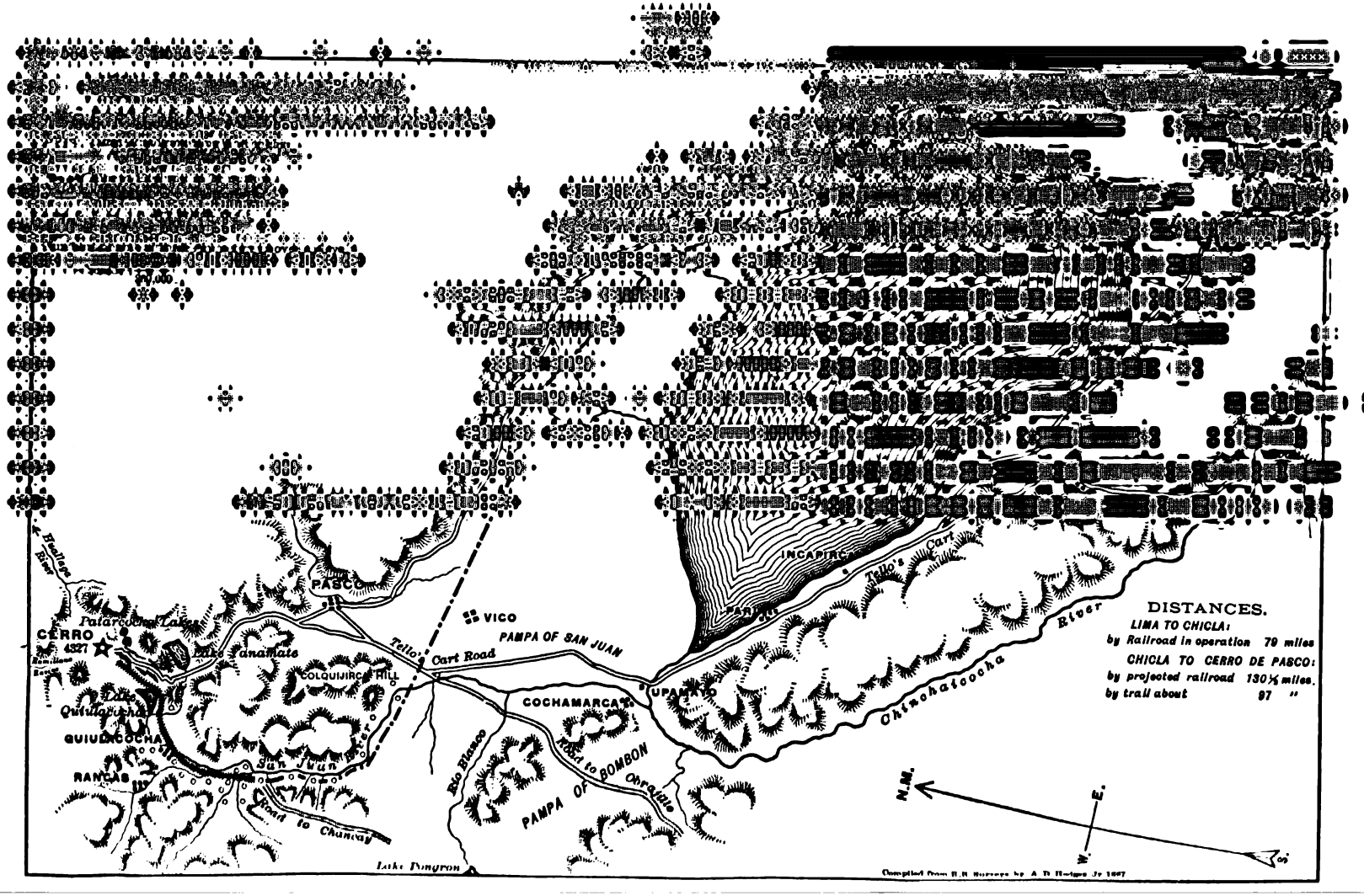
The Plateau of Junin.

In the Department of Junin is a large mountain plateau encircled by the high Cordilleras, which here unite to form the Knot* of Pasco (*Nudo de Pasco*). In this plateau, which extends north and south some two degrees of latitude, are the large lake of Junin, the pampa of Bombon, and the famous "Basin of the Cerro," where, in latitude 10° 45′ 45″ south and longitude 78° 24′ west of Paris, is situated Cerro de Pasco, the capital of the Department. Fig. 2 is a map of this plateau, compiled with care from accurate surveys.

Lake Junin (also called Lake Chinchaicocha and Lake of the Kings), a body of water with an area of some 200 square miles and an altitude of 13,380 feet above the sea, covers the southern and principal part of this plateau. From its northern end issues the Upamayo or Chinchaicocha river, which, commencing to flow northerly, soon bends completely around, receiving the waters of the San Juan and the Colorado rivers, and then flowing southerly behind the narrow range of hills bounding the lake on the west, unites with the Huamanca river near Oroya, and passes out through the southern depression of the plateau to join finally the Amazon.

In the lake are several varieties of fish, and duck and other game-birds frequent it in numbers. Along its level easterly shore is good pasturage, and the breeding of animals is carried on here to some extent. At its southern end is Junin, a small town famous as the place near which the Peruvians won a great victory over the Spanish forces in the war of Independence. There are a few villages near it. The most important locality in the vicinity is San Blas, situated about half way up its western shore in the hills which here meet the lake, and some 600 or 700 feet above its waters. Here are the salt-mine and works which supply the Cerro.

* The junctions of the eastern with the western ranges of the Andes are called *Nudos* or Knots.


DISTANCES.
LIMA TO CHICLA:
by Railroad in operation 79 miles
CHICLA TO CERRO DE PASCO:
by projected railroad 130½ miles.
by trail about 97 "
CERRO
4327
Patarcocha Lakes
Lake Yanamate
Lake Quiulacocha
QUIULACOCHA
RANCAS
San Juan River
Road to Chancay
Lake Pongron
PASCO
VICO
PAMPA OF SAN JUAN
Tello's Cart Road
COLQUIJIRCA HILL
COCHAMARCA
UPAMAYO
Rio Blanco
Road to BOMBON
PAMPA OF Chrajillo
INCAPIRCA
Tello's Cart
Chinchaicocha River
N.M.
E.
W.
S.

North of the lake is the Pampa of Bombon, the easterly division of which is often called the Pampa of San Juan.

At the northeast of the Pampa of San Juan is the old town of Pasco, now nearly deserted, but said to have been formerly (before the discovery of the mines of Cerro de Pasco) an active mining camp. Directly west of Pasco, across the Pampa of San Juan, are the hill and once famous vein and mines of Colquijirca, where evidently much work has been done in times past, but where only spasmodic efforts at mining have been made of late, the ores being sulphuretted and unsuited for the patio process.

Still further north are the hills around Cerro de Pasco, familiarly known as "The Cerro"; and at the extreme north or northeast of the plateau the Huallaga river, rising from springs near the last-named town, breaks through the Cordillera, and flows northeasterly to unite with the Amazon.

The Basin of the Cerro.

The basin of the Cerro is formed by an irregular circle of hills surrounding it on all sides. It is composed of a series of small terraced plains (partly covered by water or bogs, and rising in level from the southwest to the north), and of a low central ridge, the site of the town and of the larger part of the mines. At the southeast the encircling hills are broken through by the ravine of Quiulacocha, which is comparatively broad and contains Lake Quiulacocha, the receptacle of the drainage of the basin. The San Juan river carries the waters of this lake to the Upamayo river. At the north of the basin is the pass into Rumillana Ravine, where is located the mouth of the Rumillana tunnel; and at the northeast is the pass into Pucayaco (or Sullumarca) Ravine, where rises the Huallaga river. The latter pass is about as high as the highest point of the central ridge.

This central ridge is the Cerro (hill) de Pasco. It is about 1½ miles long (from north to south), and about three-quarters of a mile wide. On the west it is bounded by the swampy pampas of San Judas and San Andres, on the north by the rocky pampa of San Juan, on the south by the depression partly covered by the Esperanza lake, and on the southeast by the two lakes of Patarcocha. From the hills bounding the basin one spur (Pariajirca hill) slopes down to meet the ridge at the northeast, and another, narrow and low, at the south connects it with Uliachin hill. The town is laid

out on its backbone and eastern slope, while its western slope is substantially occupied by a series of immense quarries or open cuts, called *Tajos* or *Tajos abiertos*.

Mines have been worked to a greater or less extent over all parts of the ridge, as well as on some of the hills bordering on the basin. Many of the mine-openings are inside of yards in the town, some are in the streets, the majority now worked are in or around the tajos.

The altitude of the town, taken as that of the principal market-place, is 4327 meters (14,193 feet) above sea-level, varying from 4289 meters (14,072 feet) at the station of the Mineral Railway to 4354 meters (14,285 feet) at the top of Sta. Catalina hill.

The population at present is estimated at from 7000 to 8000.

The buildings are of stone and adobe, with tin or thatched roofs, and one or two stories high. Some of the houses are quite comfortable. The dwellings of the working-class consist generally of one room without window, and are heated by miserable smouldering fires of peat, and by the radiated warmth of the occupants. The better classes use the soft coal of the region in grates and stoves. The streets are in poor condition, and either roughly paved or without pavement. There are shops where the principal demands of living can be supplied, and a couple of unattractive hotels. Drinking-places are very abundant, almost all the mining bosses and all the employers (except the Railway Company) keeping rum for sale. There are several churches, the government Assay Office where the silver bars are melted and assayed, and a hospital (unfinished). At Mesapata shaft are good stone buildings and a large steam-engine with boilers and pumps in excellent condition, ready for work. The steam-pumps at the other shafts are rapidly becoming things of the past. A small establishment for lixiviation, one or two native smelting-furnaces and two or three roasting-furnaces (for manufacturing the small supply of "*magistral*" used as a reagent in the crude amalgamating process) have periodic spasms of activity. The Mineral Railway, completed and running to Sacrafamilia, 7 miles distant, connects the mines with various amalgamating works (*haciendas*), at the south. The Esperanza works, using steam-power, are prominent in the suburbs.

The most striking feature of the place is formed by the huge tajos which line the western slope of the ridge and pass into and through the town-limits, threatening its existence, as indicated by the ruined buildings around the edges of the pits. Huge cracks in the ground

adjacent to these tajos are constantly opening and perhaps closing, but attract no notice from the residents, except in the case of the special family whose dwelling commences to fall.

The tajos were formed originally by the caving of the mines. During 250 years, since 1630, the miners have been burrowing like moles under the surface, driving here and there in a most unsystematic manner, crossing and recrossing the same ground, extracting the richer ore and dumping the poorer where most convenient, and afterwards returning for this poorer ore; and all this time making no attempt to secure the ground except for the moment, or to provide for future operations. They have excavated huge chambers underground, and left them to stand or fall as might chance. An untold number have fallen; some have stood and are still to be seen, 150 to 200 feet long, 50 to 75 feet wide, 15 to 25 feet high. And labyrinths of connecting passages and chambers exist, so intricate that, without a guide, one dare not penetrate far into them for fear of getting lost.

When the Tajo Matagente first caved, 300 men, underground, are said to have perished.

A tajo, once formed, is constantly enlarged by subsequent caving, by falling of the sides and by quarrying of the walls.

The superficial area of the tajos of Sta. Rosa and Tingo (which connect) is about 41 acres, of Tajo Matagente about 9 acres, of those of Cayac about the same as the last.

As we cannot now reconstruct the original surface-lines over these large areas, it is impossible to determine, with any exactness, the amount of material removed. The present sides, sometimes formed by toppling crags, vary from a few feet to hundreds of feet in height. From the lowest point of Sta. Rosa tajo to the top of Sta. Catalina hill, which is moving into the tajo, is a measured vertical height of 329½ feet.

If, for the sake of a general estimate, we assume the average depth of the Sta. Rosa and Tingo tajos to be 100 feet over a superficial area of 1,800,000 square feet, we have 180,000,000 cubic feet, or somewhere near 9,000,000 tons, extracted at this locality alone, from vertical depths ranging up to 350, or perhaps 400 feet. The removal of any such amount has naturally resulted in constant caving and movement of the upper zones, a result intensified by periodical saturation of the mass with water during the rainy seasons. That thereby the surface-deposits have been altered to some extent in structure, appearance and relative position, will be readily understood;

also, that it is difficult at times to determine whether certain local phenomena are due to purely natural causes or to the works of man.

CLIMATE.

The climate of the Cerro has had the reputation of being terrible —a reputation due partly to the reports of persons accustomed to the warmth of other parts of Peru, who visited the place before coal was used here, and partly to *a priori* deductions from the altitude. This reputation is undeserved. Extremes of cold and heat are unknown. From August, 1886, to March, 1887, the temperature (taken with a maximum-and-minimum thermometer) ranged from 28° to 64° Fah. The lowest point which I have seen recorded is 10° Fah., in the night (Raimondi). At San Juan, about 10 miles from the Cerro in a straight line, and some 800 feet lower, observations taken during one year (1877–78) gave an *average* of 40° Fah. July, August, and September are the coldest months—December to March the warmest ones.

To one out of doors, the temperature often seems to vary greatly in a few moments; it being, for instance, at midday in full sunshine very warm, and five minutes thereafter, when a passing cloud obscures the sun, quite cool. Hailstorms, or snows, or rains, are liable to occur at almost any time, especially during certain seasons of the year, and are very disagreeable. The frosts at night are bad for agricultural crops. But more than two or three inches of snow on the ground, or more than a mere skim of ice on pools of still water, is an exceptional occurrence, and both snow and ice disappear almost immediately under the sun.

Rains are prevalent, as a rule, from November to March, being most abundant in the first three months of each year. From March to November rain is exceptional, but hailstorms and snowstorms occur now and then. The rainy season is warmer than the dry season, but fires are needed for comfort during the whole year.

On account of the altitude and consequent thinness of the air, new-comers are troubled with shortness of breath after any considerable exertion. But the air is tonic and bracing, and after rapid inspiration I always experienced an exhilarating effect. There is no special danger from this cause at the altitude of the Cerro; but at elevations of 17,000 feet, or more, it is unsafe to urge animals or men rapidly up hill. Occasionally an animal drops dead under such circumstances. The consequences to human beings are apt to be

nausea and headache (*siroche*), which may be felt at much lower altitudes, but which, according to my experience, are due chiefly to improperly loading the stomach, particularly with beer and liquor. I have repeatedly crossed the mountains at heights of 17,000 feet, with from two to a dozen companions, and never were any disagreeable emotions felt when my advice as to food, drink, and rate of progress was followed.

There is but little sickness at the Cerro. All the cases which have come to my observation were to be attributed, not to the climate, but to evil habits of living, principally intemperance. Colds, however, are common.

There are foreigners now living at the Cerro, apparently in an excellent state of health, who have resided there from ten to forty years. I have had men work hard for me, frequently as much as eighteen hours a day, without trouble. The custom of the country, however, is opposed to long hours of work—in fact, it is opposed to any work at all.

There is remarkably little wind at the place. Anything more than a mild breeze is an exception, although a strong wind occasionally occurs.

To sum up the subject, I regard the climate of the Cerro as unusually wholesome for those having proper conveniences of life, and plenty of warmth and ventilation, but disagreeable and trying to some constitutions; and I should advise against long-continued residence without occasional changes to warmer regions.

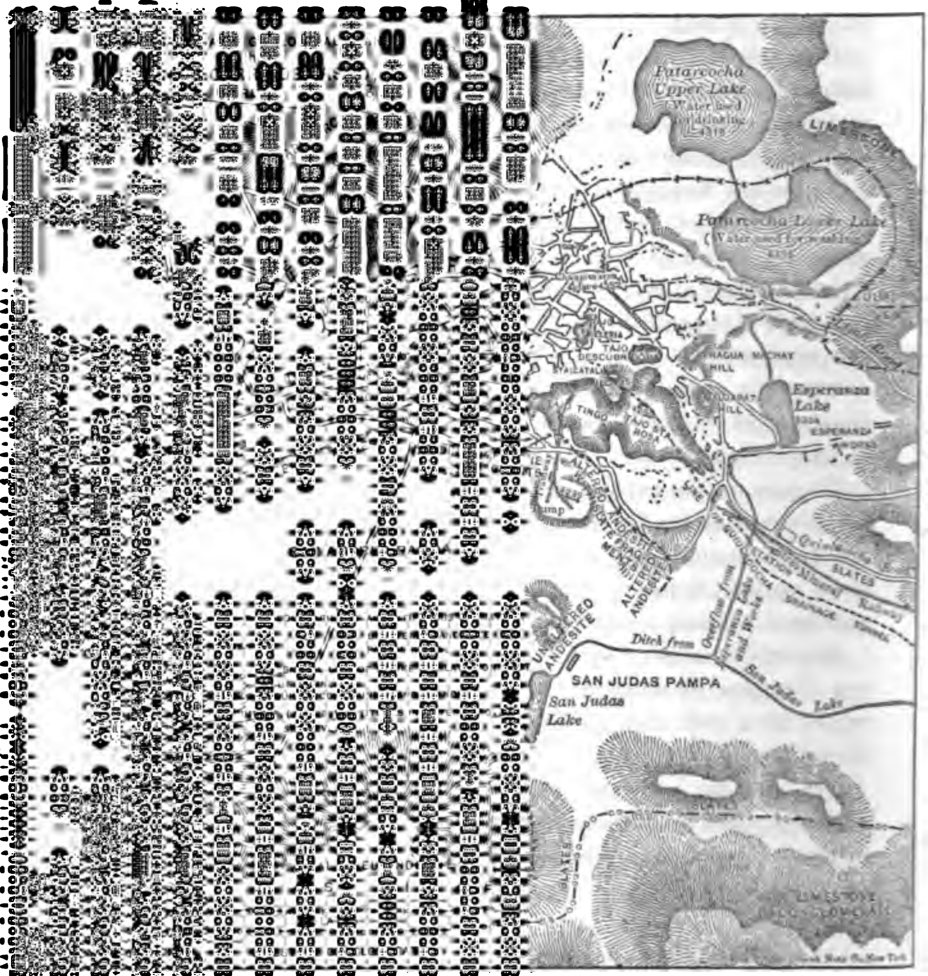
But at distances of eight or ten miles from the Cerro, in almost any direction, by descending the steep ravines, one can reach places where is a soft and pleasant climate, and where the vegetation is abundant and beautiful.

The climate is too cold for agricultural crops. Barley grows, but does not ripen. The pampas supply abundant grass for sheep and cattle. Vegetables are raised in the ravines at lower altitudes.

Physical Aspect.

Fig. 3 is a general topographical and geological plan of the basin of the Cerro. The whole region around the Cerro at first sight is apt to appear dispiriting. The trails are rough. Barren hills of limestone, slate and sandstone, often in strangely-contorted or sharply-tilted strata of Jurassic and Cretaceous age, rise abruptly on all sides. The pampas are rolling, generally covered with short green grass and,

in treacherous bogs where lives. Here and there are rds, near rows of low stone- rds of cattle, or, perhaps, of



but the landscape is often villages—collections of low, s, which run in all directions the amalgamating-*haciendas*, ving or quiescent millstones,

circular stone-enclosed *circos*, rows of one-story stone huts with thatched roofs, and perhaps a miserable chapel. The sky is generally leaden-gray, the air is apt to be chilly, and pelting rain or hail causes much discomfort. The accommodations for the traveler at night are rude and cheerless, and the food is perhaps scanty, perhaps cooked over a fire of *tachia* (llama dung). Yet with bright sunshine, the snow-capped Sierra on the horizon may look most beautiful, and the cloud-effects at times are very grand. Short-stemmed flowers adorn the pampas many months of the year, and the region abounds in deep ravines where one unexpectedly drops down a couple of thousand feet into wonderful scenery, a balmy atmosphere and a profusion of trees and flowers, and of fruits and vegetables in their season. Seldom have I experienced a pleasanter abrupt change of climate and landscape than when, on the first day of January, 1887, I passed from the dreary heights above Chacayan down into the gorge of the Vilcos river, whose steep sides from top to bottom, a sheer vertical distance of two thousand feet, were completely covered with the terraced potato-fields in full bloom, and showed only the green of the leaves and the whites, reds, blues and violets of the blossoms, contrasted with the bright yellow of intergrowing weeds; where roses, buttercups single and double, dahlias, marigolds, violets two inches in diameter, heliotropes, geraniums and numberless other plants were all arrayed in their most gorgeous garments, and ferns of exquisite form and texture adorned the rocks and caves. Until I visited the Peruvian Andes, I never knew how to grow enthusiastic over a potato-field.

In the vicinity of the Cerro, the numerous mining-*haciendas*, sometimes perched in unexpected places, form quite a feature of the landscape. Wherever water to run the stones can be obtained, even if the supply is for only a few months in the year, there the *ingenios* have been erected. According to a rough estimate, the combined grinding capacity of all these *haciendas* is about 185,000 tons yearly; but many are rarely or never used, and some are falling in ruins.

Geology.

The mining belt of Peru is made up of rocks of Jurassic and Cretaceous age.

In and around the basin of the Cerro there are visible on the surface, limestone conglomerates, limestones, andesites, slates and sandstones and the argentiferous formation. Fig. 3 shows their

relative positions. Fig. 4 is a general section across the basin from east to west, so drawn as to include the main elements of the rock series. If the line of this section were traced on Fig. 3, it would run from Pariajirca hill southwesterly to the center of Tajo Santa Rosa, and thence northwesterly to Pargas ridge.

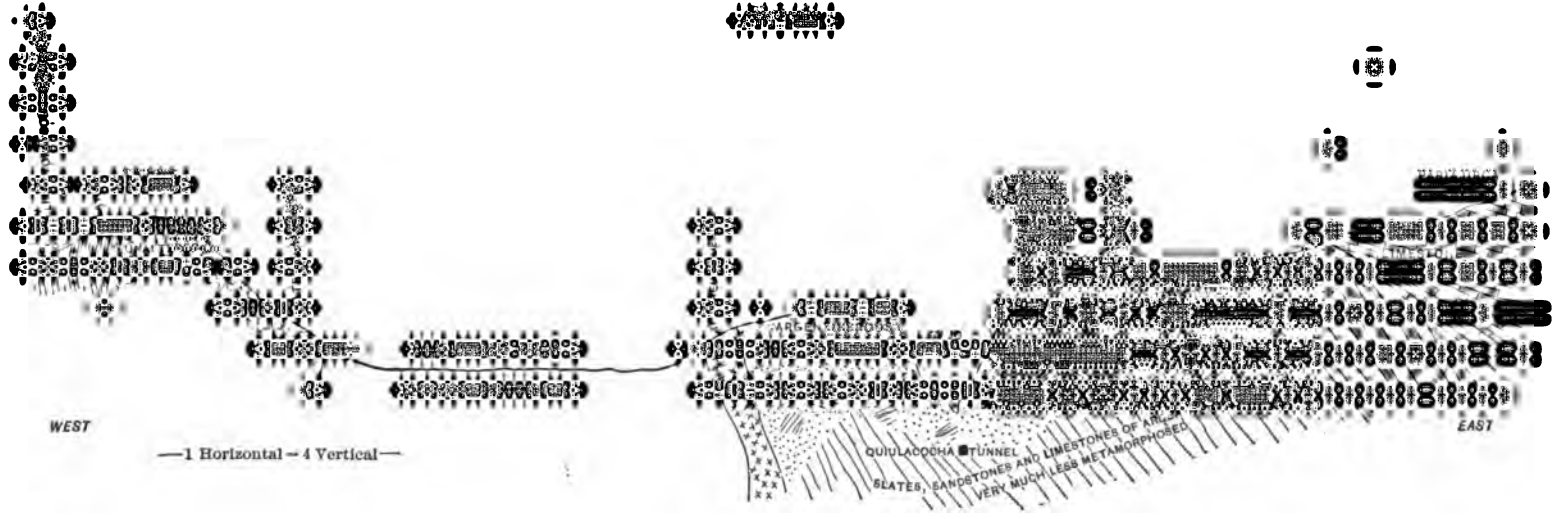
1. *Limestone Conglomerate.* A coarse limestone-conglomerate forms the extreme western boundary of the Basin, occurring on the ridge west of San Andres and San Judas pampas and capping the peaks on the south where it is visible both on the high hill west of Lake Quiulacocha and also on the east of this lake from Piedras Gordas (on the Mineral Railway) to Uliachin. This is the newest of the sedimentary rocks and is uncomformable with the underlying strata.

2. *Limestones* bound the argentiferous formation on the north and east and in part on the south. They form the hills surrounding the Basin on these sides and extend northerly and easterly for many miles. They vary in color and structure—being black, white, bluish or yellow, and reddish from iron-stains; and fine-grained to coarsely granular. The strata are nearly vertical at one point on the west side of Pucayaco (Sullumarca) ravine, north of the Basin, but are much less inclined further south. Rivero states that there are vertical strata of blue limestone in Uliachin hill (at the south), which correspond with those at the north.

In certain places, as around Uliachin, fossils are comparatively abundant. Raimondi mentions *Encrinites* and a few rare specimens of *Neithea quinque-costata*, which indicate that the formation belongs to the Cretaceous epoch.

The strata apparently vary somewhat in age, not always being conformable with one another or with the (generally) older strata of slates and sandstones.

In the limestone are numerous veins. Those at Uliachin contain sulphurets of varying richness, have a general north and south strike, and abound in mine openings which extend nearly or quite to the town ridge. They were supposed by Trevethick to continue south for miles (to Colquijirca and beyond) and form his "Colquijirca lode;" but between Uliachin and Colquijirca, which is five or six miles distant, no evidences of this lode are visible. On Pariajirca hill are several veins, of which the one where the Concordia and Negrillos mines are situated, strikes easterly and westerly, dips nearly vertically, has a width of six to ten feet for several hundred feet of length opened, and holds rich tetrahedrite and other sul-



GENERAL SECTION ACROSS THE BASIN OF THE CERRO DE PASCO, PERU.

phurets. On and around this hill and on San Juan pampa are numerous mines in limestone, some of huge size, but all except two or three now abandoned. Just north of the cascajo, on San Juan pampa, and not far from the andesite, the limestone in depth, in certain places at least, is very siliceous, abounds in cavities, and holds more or less pyritic material. On the east of Rumillana ravine, in San Juan hill, are old workings which indicate veins in the limestone, but which at present show nothing of interest.

3. *Eruptive Andesites.* An unaltered coarsely-granular mass of quartz-biotite-andesite occurs on the hills of Pargas ridge, west of San Andres, and is traceable here from a point opposite San Judas lake northerly to the Tucsi road and possibly beyond. On the eastern slope of these hills, at the eastern end of San Judas lake, at the railroad quarry near the station, and at numerous points along the western boundary of the argentiferous formation, are dikes of rock essentially the same as that on Pargas ridge, but finer-grained and more decomposed. The bogs of the San Andres and San Judas pampas prevent exploration, but boulders or projections of similar appearance are visible at a few points along the trails across them.

This rock (as determined by Mr. F. J. H. Merrill, of the School of Mines, Columbia College) is composed essentially of distinct crystals of plagioclase and biotite, with irregular and generally very small grains of quartz imbedded in a felsitic base. The biotite is dense and opaque, and, judging from its decomposition, rich in iron. There occur, as accessory minerals, small crystals of orthoclase and apatite and some grains of magnetite. Noteworthy is the fact that crystals of pyrite are found abundant in some places but are entirely wanting in others. The rock in certain localities is a good building stone, and an excellent fire-proof material when free from pyrite.

The dike next to the argentiferous formation contains, south of the tramway-cut into the Tajo Tingo, fragments of sedimentary rock (slates), and has been considerably metamorphosed, probably by the action of thermal waters, which have removed the iron from the mica, kaolinized the feldspar and almost wholly obliterated the structure of the rock. Northerly of the Tajo Tingo, what is, to all appearances, the same dike holds angular fragments of unaltered limestone which, going south, gradually decrease in size until entirely wanting, being replaced by fragments of slate. The surfaces of this dike-rock, after long exposure, acquire a blue or bluish-gray tinge. Fresh fractures are grayish or often pinkish.

The eastern boundary of this dike can be traced on the surface along the argentiferous formation from near Rumillana pass (at the north) into Ayapoto district (at the southwest), where it seems to swing around westerly, near San Judas Lake, towards the southern extremity of the Pargas Ridge andesite. The western boundary is obscured by the bogs of San Andres pampa. It is possible that I may have included several dikes under this one.

The interposition of the swamps of San Andres and San Judas pampas (where three times I nearly lost my riding-mule in attempts at exploration) unfortunately has prevented the determination of the rocks of a large and important area, and makes difficult the verification of several geological points. Some miners assert, and many believe, that there is mineral ground here; and at one point southwest of the dump from the Tajo Tingo, and therefore apparently west of the eastern andesite dike, there is pyritic material, which was being prospected at the time of my visit.

In order to get as much light as possible concerning the eruptive rocks, I submitted a few specimens to the careful microscopic examination of Mr. John H. Caswell,* of New York. Mr. Caswell's results confirm the statement that all the rocks are quartz-biotite-andesite, and that their testimony, so far as it goes, is not unfavorable to the theory of the existence of dikes of varying ages.

Mr. Caswell's determinations were as follows:

(*a.*) Specimen from Pargas Ridge.—"Gray, light-colored rock, containing large whitish feldspars and black biotites, and quite fresh in appearance and comparatively unaltered. It is a quartz-biotite-andesite, containing fresh clear plagioclase crystals, large pellucid quartzes and clear brown biotites in a very fine-grained trachytic ground-mass.

"The plagioclases have the characteristic sharp twin banding and show also a zonal structure. The quartz has often a crystal outline and inclusions of portions of the ground-mass. The biotite is clear

* I gladly avail myself of this opportunity to refer to the unusual conscientiousness and large mineralogical knowledge of Mr. Caswell, whose sole defect—that of modesty—has alone prevented more general recognition of his ability. His monograph on the *Microscopic Petrography of the Black Hills of Dakota* (U. S. Geog. and Geol. Survey of the Rocky Mountain Region) was written in 1876, and was one of the earliest works of the kind in this country. Unfortunately it was not published by the Government until 1880. The plates illustrating it are noteworthy as being exact reproductions of the microscopical sections, and, so far as I can learn, were the first published in the United States which were not combinations of, or generalizations from, a number of sections.

and very dichroitic, and has inclusions of apatite needles. As small crystals it is an inclusion in the plagioclase. The ground-mass is made up of minute feldspar crystals and some magnetite.

"Accessory are apatite in fresh large clear crystals, often doubly terminated; a few altered greenish crystals of amphibole, with the usual cleavage lines; magnetite in abundant small black grains and occasional large crystals; and titanite as pale brownish crystals, with dark opaque border or outline.

"*No pyrite* was observed."

(*b*.) Specimen from the eastern dike at a point just west of the Tajo Tingo: "Whitish rock, considerably altered, in which can be seen opaque white feldspars, *some pyrite*, and included fragments of a gray rock [slate] and quartz. It is an altered quartz-biotite-andesite.

"The plagioclases are quite cloudy and opaque, so that the twin banding can be but faintly made out. The biotite has been changed to pale-colored, skeleton-like crystals, which polarize but faintly. The quartz occurs in large pellucid crystals and also in aggregations of small grains. The ground-mass is feldspathic but much kaolinized and decomposed.

"Accessory are yellow pyrite grains; a very little black magnetite; apatite in large clear crystals and needles in the feldspars and ground-mass; and occasional skeleton remains of amphibole crystals."

(*c*.) Specimen from near the last; either the same dike or one immediately adjacent, no boundary lines being traceable: "Pinkish rock, in which can be seen glassy feldspar and black biotite crystals. It is a quartz-biotite-andesite, but a little different in character from (*a*).

"The plagioclases are only fairly clear, although the twin banding is distinct. The quartz is seen in occasional large angular fragments and also in aggregations of small grains, filling the interstices, as in (*b*) and (*d*). The biotite has a black opaque border, being translucent only in the center of the fresher prismatic crystals and then is of a dark-brown color. It has inclusions of apatite crystals. The ground-mass is feldspathic and shows a slight fluidal structure.

"Accessory are large clear apatite crystals as well as acicular ones; amphibole in a few much altered crystals; some titanite with dark borders; and abundant magnetite in black grains, scattered through the rock.

"*No pyrite* was observed."

(*d.*) Specimen taken about 300 yards south of samples (*b*) and (*c*): "Fresh yellowish rock, in which large black biotites and glassy feldspars and quartz are plainly seen. It is a quartz-biotite-andesite, with transparent plagioclases, large quartzes and biotites porphyritic in a fine-grained ground-mass.

"The plagioclases are sharply banded and have also a zonal structure. The quartz occurs in large crystals and also in small grains surrounding the altered biotites and filling many interstices in the ground-mass. The biotites are quite fresh and strongly dichroitic in the sections parallel to the prism.

"Accessory are apatite crystals in hexagonal, basal and rectangular prismatic sections, some dark-bordered; titanites; and an occasional black grain of magnetite in a biotite crystal."

"Sections (*a*), (*b*) and (*d*) appear to have similar structure in general characters, (*b*) being much decomposed. Section (*c*) has a slightly different appearance, the biotites especially being very dark and opaque, even in the thinnest sections."

4. *Barren slates*, in nearly vertical strata, bound on the west the eruptive formation on Pargas ridge, occurring between the andesite and the limestone-conglomerate, or below the last. The strata can be traced the whole length of the basin on the western hills. They strike north and south, except opposite San Judas lake, where they run east and west, as if bent around by the intrusion of the adjacent andesite.

Other barren slates, in highly tilted strata, are to be seen at the north in Rumillana ravine; also at the south at various points from Ayapoto district to the Esperanza works, and on the Mineral Railway, as well as in the Quiulacocha tunnel from Ayapoto district to its mouth. It seems quite possible that these slates are continuations of the argentiferous formation. With them occur some sandstones.

5. *The argentiferous formation* lies between the limestones on the east and the andesites on the west, and forms the central ridge on which the town is built. At the north it rapidly narrows, as shown on the plan, while at the south its boundary is undefined, there being no exploratory work here, owing in great part to the nature of the surface of the ground. Its area developed may be roughly stated as about one and a half miles from north to south by three-quarters of a mile from east to west. It is very fully exposed for a maximum depth of 300 feet by the mines and *tajos* along the back-

bone and western slope of the ridge. Elsewhere it is at present imperfectly open to inspection.

This formation has long been a geological puzzle. The present attempt at a partial solution of the problem differs from all preceding theories on the subject principally in the respect that it combines in one formation rocks which have heretofore been considered radically different.

In addition to the natural difficulties, the mining operations, while exposing the material below the surface, have complicated matters to quite a considerable extent by causing artificial movements of the upper portions of the deposit.

For convenience, and following a current phraseology, the argentiferous formation may be divided into "surface-deposits" or ground above the water-level, and "deep deposits" or ground below the water-level; and the dividing lines between these two classes may be taken roughly as occurring near the general level of the Quiulacocha tunnel, which, driven with varying grades and now caved in places, imperfectly drains many of the mines. This tunnel does not reach the northern or eastern districts, and the "deep deposits" occasionally rise nearly to the surface.

Above the water-level, the formation consists of a highly metamorphosed and greatly oxidized material, of so constantly varying structure, color and composition that it is difficult to give an adequate idea of it without a long and detailed description. Over a large portion of the town-ridge there is a hard, compact, reddish or yellowish and very quartzose cap-rock of ever-changing thickness. Below this, as a rule, the formation is softer and more decomposed, being sometimes broken into loose or cemented fragments of all sizes, and passing by all gradations of structure and hardness, but without any evident regularity, into earthy masses or soft clays or sugary sands. The smaller fragments, whether loose or cemented, are often so arranged as to present a slate-like appearance. A hard gray quartzite is frequent; porous material resembling scoria is met now and then; and a rotten slate, generally pyritic, is not uncommon. Local evidences of stratification may be seen, but generally on a limited scale, and every thing of this kind is irregular and indistinct. The predominant colors are reds, browns and yellows, due to the presence in greater or less amount of the oxides of iron, and the ore frequently resembles limonite; but grays, whites and even blacks occur, while the cupriferous portions show all the copper stains. The rock is everywhere very siliceous, always yielded a

considerable percentage of slimes when crushed wet, and everywhere contains at least traces of silver, of pyrites and of carbonate of lead (and of lime). Very rarely is the silver visible, even with the aid of the magnifying glass, and then principally in small native scales in connection with quartzite. With the exception of cerussite, of some of the pyrites and of the products of decomposition now going on, crystallized minerals are practically unknown. Quite a number of imperfect fossil shells, including specimens of *Trigonia*, were obtained by us from mines in or near Sta. Rosa district. These formed a constituent of the brown earthy cascajo. As they are a great local rarity, only an exceedingly small number having been found previously, and as their occurrence in the ore seemed particularly interesting, we spent much time, some money and considerable trouble in collecting and transporting them. They were sent as a present and for identification to the geological department of the School of Mines, Columbia College, where the identifying process appears to be a slow one.

It is noteworthy that the decomposition of the mineral constituents does not always proceed gradually from the present surface downwards. Very hard and very soft rocks often adjoin, and large bodies of solid pyrites in a chalcedonic matrix are found at varying depths, and generally in close proximity to greatly oxidized material.

The rock along the western slope, more especially in and around Sta. Rosa tajo, is rather harder, more quartzose, more ferruginous, and on the whole more uniform in its silver contents than elsewhere, and forms what may be called the "cascajo zone," the term *cascajo* being applied broadly to those ores which are worked by the *patio* process. East of this zone the ground seems to be lighter-colored, more clayey and variable, and the silver to be more concentrated in pockets. The percentage of lead carbonate seems to be somewhat greater, and coppery ores are said to be more frequent. But the formation is very much richer in copper in the district southerly and southwesterly of Yanacancha church than in any equal area (and here are large deposits); and the lead-ores (that is, those in which lead is a prominent constituent) occur chiefly at the northeast in Matagente district, where they appear to be partly overlaid by a stratum of cascajo.

The chemical composition of the surface-ores naturally varies greatly. Of the many analyses made, I select the following as representing large masses of cascajo in Santa Rosa tajo:

	Per cent.
Silica,	72.00
Alumina,	6.50
Iron peroxide,	13.50
Iron protoxide,	0.50
Iron sulphide (FeS_2)	2.00
Lead carbonate	1.25
Lime (and magnesia) carbonate,	1.50
Manganese peroxide,	0.55
Zinc (combination undetermined),	0.40
Copper "	0.05
Antimony "	0.25
Arsenic "	trace
Sulphur* "	0.30
Silver,	from traces upwards

In this connection I insert the following analysis of a hard pyritic ore from the deeper deposits:

	Per cent.
Silica, etc.,	40.05
Iron,	26.63
Copper,	2.73
Nickel,	trace
Silver,	0.13
Sulphur,	26.55
Antimony,	2.40
Arsenic,	trace
Moisture,	0.95
	99.44

Gold occurs in the merest traces, and thallium has been detected in the bullion.

The condition of the silver has not yet been satisfactorily determined. All direct tests for chlorine have given negative results. A part of the metal is unquestionably in a metallic state, as may be seen occasionally. Undoubtedly it exists in varying combinations in the different classes of ores. I judge that in the cascajos from 50 per cent. of the silver in the harder varieties to 85 per cent. in the softer material is metallic, and that the remainder is combined with sulphides or antimonides, or with lead carbonate. Argentite, often earthy and decomposed, and sternbergite, occur now and then.

The pyrites are sometimes nearly pure iron pyrite, sometimes

* Existing as SO_3, according to Prof. Raimondi's analyses.

chalcopyrite, rarely bornite and mispickel, often a mixture of two or more of these varieties. Pure pyrite is usually poor in silver, but not always. The pyritic material is often associated with silver sulphides, and passes into tetrahedrite, which occasionally is very rich.

The sulphurets of silver, copper and iron just mentioned are common to the formation both above and below the water-level, and to the veins in the limestone. Rich masses occur in pockets of greater or less size and irregularly disposed, and their occasional discovery is the great stimulus to the exertions of the miners.

Native copper occurs very rarely (as in the Calepe mine) in arborescent or rounded lumps and plates of small size. Amorphous cuprite, malachite and azurite are found, also bournonite, and various sulphides of copper, or of copper, iron and lead. Crystallized sulphate of copper, generally contaminated more or less with iron sulphate, is seen very often, sometimes in large and beautiful specimens with curiously contorted crystals, as in the San Jose de Gaona and Toril mines.

Zinc is reported in all analyses. I noted it in place only in the deeper deposits (in the rare shape of zincblende) and in the common efflorescences on the sides of the drifts.

Galena, at times very rich in silver, is found in bunches, chiefly (so far as I know) southerly of the large copper-deposits and in Matagente district.

The preceding description applies to the material *above* the water-level. Below this level, the ground is at present accessible only along the Quiulacocha tunnel and its branches, and in the very few deep mines now open. We have, however, the oral testimony of the older miners, the written description of Rivero, and the results of a few deep drill-holes. This evidence is sufficient to show that under the highly altered surface-rock there are slates, sandstones, and limestones, in strata which (according to Rivero), like every thing east of the andesite, have a general northerly and southerly strike and an easterly dip; which contain quartz, calcspar, pyrite, and chalcopyrite very generally, and often in high percentages; and in which rich deposits of sulphurets and occasional native silver have been found in times past.

The rocks in question, although as a whole very much less altered than the surface-material, are, nevertheless, metamorphosed to a considerable extent.

The slates greatly predominate in the parts of the ground which has been explored. A typical gray slate, quite soft, and traversed by veinlets of quartz, calcspar, and pyrites, occurs in places. More commonly visible is a black slate, fine-grained and frequently much broken, often very hard, sometimes strongly silicified, holding at times particles of mica and small seams of white quartz or quartz and calcspar, and containing pyrites in very-varying quantities with occasional bunches of tetrahedrite, and (rarely) zincblende. Pyrites are so common and so abundant that the miners apply the name of "pyritic formation" (*bronce* or *bronce-formacion*) very generally to this rock. The slate passes into the rotten pyritic variety common in the surface-ores, or (apparently) into the dark barren slates to the south, or again into what I can best describe as a siliceous sedimentary rock—a fine-grained, stratified material, in which silica appears to have replaced the original constituents. There occur, also, very hard quartzose masses and large bodies consisting of a chalcedonic matrix full of pyrites. This last rock can be seen, not only below ground, but also above—notably in the Tajo Tingo.

The limestones, so far as I know, are very limited in quantity. I observed them in only a few places, as at the junction of the San Pedro de Cardenas drift with the branch of the Quiulacocha tunnel, where quite a large body of magnesian limestone is visible. I am disposed to refer the limestone in depth in the San Juan pampa near the cascajo boundary to this formation. The occurrence of limestone fragments in the eastern andesite dike indicates the existence of the rock in depth.

The top of this less altered deposit, which seems to underlie all the surface-formation, rises and falls in undulations, in some places sinking deep below the Quiulacocha tunnel, in others coming up nearly to the present surface of the ground.

The transition from the highly metamorphosed rock above into the plainly sedimentary rock below is often very sudden. But nowhere is there visible any clearly-defined contact-plane. On the contrary, the contact-lines are very broken, run at all angles, and are traceable individually only for a few feet. That is, they are such as would be formed by the folding and breaking of the strata. Occasionally, cascajo is found *below* the less altered rock in bodies of unascertained extent.

Very commonly there is an intermediate zone, where the decomposition of the sulphides and their conversion from the conditions usual in the deep deposits into sulphates, carbonates, oxides, and

very likely other combinations, may be seen in all grades, and on a very large scale. The waters of the Quiulacocha tunnel are very heavily charged with acid sulphates, and the walls of the drifts are often covered with efflorescences of varied chemical composition.

Concerning the argentiferous formation there are different theories.

Trevethick, one of the greatest mechanical geniuses of the world, who was at this place from 1813 to 1819, and put up the first steam-pumps, proposed the idea of two great veins—that of Colquijirca coming in from the south, and that of Pariajirca from the northeast, the two uniting under the town and then swinging off as one lode to the northwest, with a total length of some seventy miles. This theory, formed when the developments were very limited, is mentioned here simply as a matter of record, and because it is the one still generally published in the text-books.

A common idea among the miners is that there are three broad, parallel, and nearly vertical veins running northerly and southerly —the vein to the west containing cascajo ores; the one in the middle, coppery ores, chiefly in pockets in clayey ground; and that on the east, lead-ores. The boundaries between these veins I have been unable to trace further than as indicated previously. Some miners deny the veins, and prefer the theory of beds.

Rivero, an acute observer, who was engaged in directing the building of the Quiulacocha tunnel, in 1827 and 1828, was the first to solve (partially) the problem of the cascajo formation, which he determined as a highly metamorphosed sedimentary rock, and which he took to be sandstone. His theory was, that the cascajo zone is a "*masa echada (liegender Stock)*," underlain by a pyritic slate formation; and that the silver was derived from the decomposition of the pyrites. Rivero had access to many of the deeper workings which are now filled up. His *Memoria sobre el Rico Mineral de Pasco*, written in 1828, is, in many respects, the best and fullest description yet published of the place.

Professor A. Raimondi, in his brief *Memoria sobre el Cerro de Pasco* (Lima, 1885), finds that the three formations of beds, pockets and veins occur. "The chief argentiferous bodies at Sta. Rosa and adjacent tajos," he writes, "form a large pocket (*bolsonada*) where the silver is disseminated everywhere, forming the immense deposit of cascajo." He agrees with Rivero that the cascajo is a highly altered sandstone. This sandstone formation, he thinks, has been tilted and broken by a diorite "which is not visible on the surface,

but which, without doubt, occurs at a certain depth, there having been found here and there various pieces of dioritic rock composed of feldspar and amphibole. It is very probable," he adds, "that here, as in many mining districts in Peru, the dioritic rock has raised the strata of sandstone and limestone, and introduced the metalliferous mass into them, without appearing on the surface, and that the trachyte [the rock which I call andesite] was a later eruption."

That the surface-ores are in part a metamorphosed sandstone may, I think, be considered an established fact. But to my mind it is extremely probable that a very large proportion of the siliceous rocks holding alumina, and existing both in solid masses and in the shape of the abundant clays, has been formed by the metamorphism of slates; indeed, as before stated, slates only partially altered and perfectly recognizable are by no means uncommon. I have not met with limestone in mass near the surface; but carbonate of lime and other carbonates are of universal occurrence in at least small percentages. After a long study of the ground, I have been led to the conclusion that the surface-rocks and the deep deposits are made up of essentially the same materials and differ chiefly in the degree and kind of metamorphism which they have undergone.

My impression is, that the site of the present Cerro was once covered with strata (more or less horizontal) of slates and sandstones and, to a certain extent, limestones, which now form essentially what I call the argentiferous formation; that these strata have been repeatedly tilted, the western portions being gradually raised until they came to or above the present surface-line, and naturally are more broken and altered than the easterly parts which now occupy levels below them; that there have been various eruptions of andesites, which rocks are now visible on the west of the argentiferous formation on both sides of San Andres pampa; that, accompanying or following these eruptions, there have been ejected from below siliceous and metalliferous solutions which have attacked most strongly the more broken portions of the strata, impregnating them with silica and silver and other metals, and otherwise altering them, such metamorphism being reinforced by subsequent exposure to atmospheric influences and intensified by succeeding eruptions of the andesite; that the limestones at the east were deposited before the time of the latest upheavals and impregnations which tilted and cracked them, and formed and filled with ore the veins now seen in them; and that the last period of the geological history was that of

the final weathering and erosion which gave the surface-rocks their present outlines and appearance, and of the deposition of the limestone conglomerate visible at the south and west.

THE BLAKE SYSTEM OF FINE CRUSHING AND ITS ECONOMIC RESULTS.

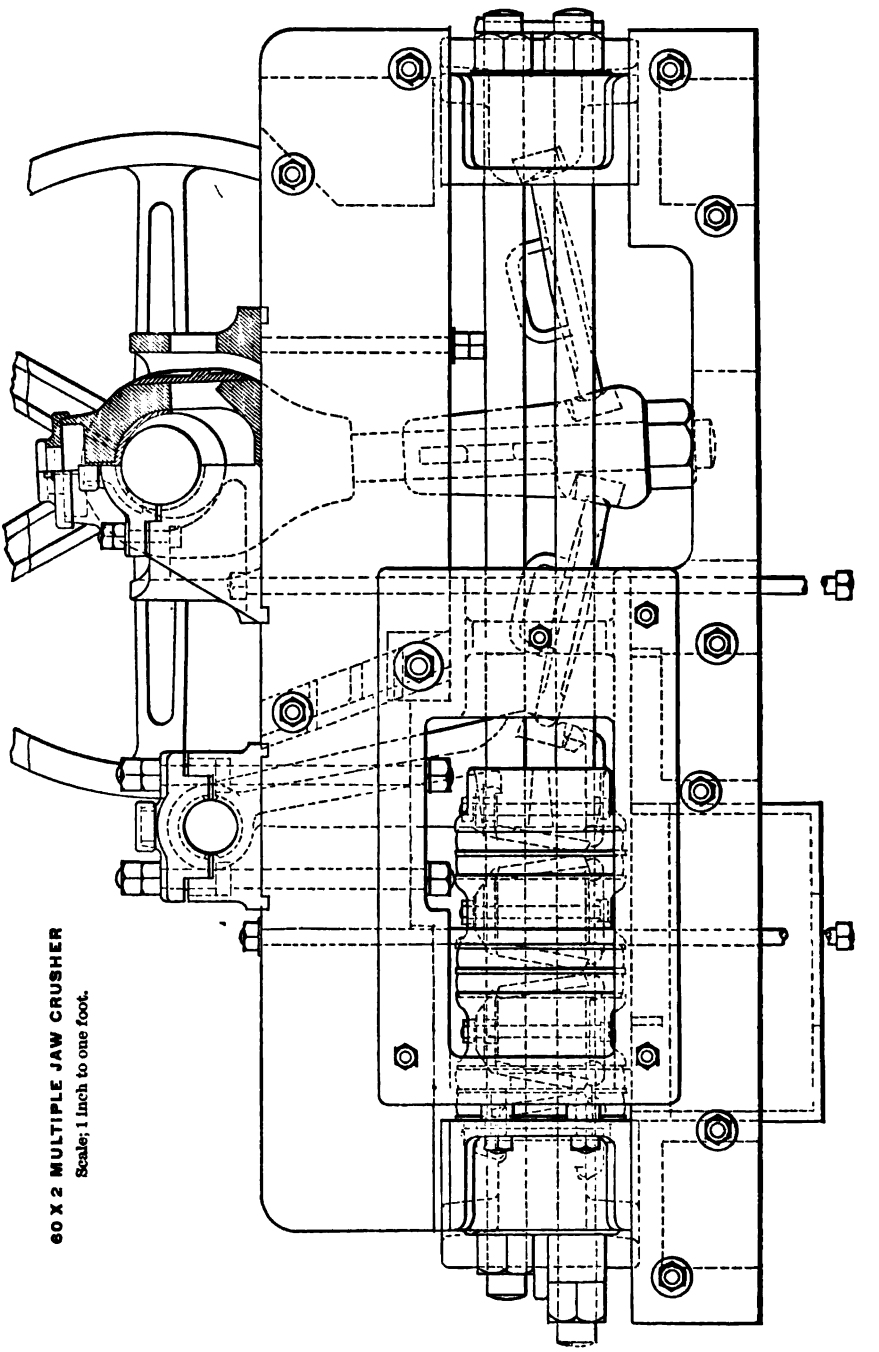
BY THEODORE A. BLAKE, NEW HAVEN, CONN.

(Boston Meeting, February, 1888.)

AT the Chicago meeting of the Institute, May, 1884, I had the pleasure of announcing the introduction of a new machine for fine crushing, or The Blake multiple-jaw crusher, which, in combination with the ordinary Blake breaker, could be used in the reduction of ores or any hard and brittle substance to almost any degree of fineness. I am now able to give some practical results derived from its use on an extensive scale, on the gold ores of South Carolina and the iron ores of Lake Champlain, together with some accurate and detailed statistics of the cost of crushing over one hundred thousand long tons of magnetic iron ores of the Chateaugay Ore and Iron Company at Lyon Mountain, Clinton County, N. Y., about thirty miles west of Plattsburgh.

The construction of the multiple jaw-crusher, since the date of my first paper, is the same, except in the substitution of main swinging, instead of main sliding, or toggle jaw,—thus doing away with the upward thrust on the tension rods and the wear incident thereunto. (See Fig. 1.) It has also been found, in case of the fine crusher, that is, in machines of not over ½ inch width of opening, that the use of several small machines with a series of jaw openings, say 15 inches by ½ inch, is better than one large machine with a series of openings 24 by ½ inch, or 36 inches by ½ inch, as at first constructed. Many details of the method of holding in jaw-plates, etc., have been perfected.

Without giving any details with respect to the different mills built upon the Blake system, let me state that, although it has proved *possible* to carry any hard and brittle material with crushers alone to a fineness such that all particles will pass a 30-mesh wire screen, still the economical limit of such crushing will be found somewhere between 14 and 20. If it is necessary to carry all the material to a fineness greater than this, the system must be supple-



60 X 2 MULTIPLE JAW CRUSHER
Scale; 1 Inch to one foot.

Fig. 1.

mented with other arrangements. Furthermore, the material to be operated upon must be sufficiently dry to screen readily, to take out the fine as rapidly as made, or fed with such an excess of water as will insure successful screening. It is, of course, evident that if fines are not removed as rapidly as made, there must be an accumulation and consequent slowing down of feed, and greatly diminished product, or a stoppage.

The first mill on the Blake system was that built for the Chateaugay Ore and Iron Company in 1882, which was designed to crush two hundred tons per day down to pass a quarter inch round hole. This was run at irregular intervals, and idle much of the time up to the summer of 1886, owing to the depression of prices of iron ore. Since the summer of 1886 it has been running continuously, crushing many thousands of tons of the tough magnetic ores of that company. I am not informed as to the exact product and cost of operations of this mill, but they have been approximately the same as that of a mill of treble the capacity built for the same company on the same general plan in the summer of 1886, for exact and accurate details respecting the operations of which I am indebted to the Chateaugay Ore and Iron Company, and enabled to give statistics which will be found in another part of this paper.

The second mill of any considerable magnitude is that of the Haile Gold-Mining Company, Lancaster County, South Carolina. The gold-ores derived from the different veins or mines upon the property are altered magnesian slates. That of the "Blauvelt" mine is a very tough, coarse quartzite, carrying a large percentage of sulphurets. Free gold is rarely, if ever, found in these mines. For purposes of amalgamation, the ore must be crushed extremely fine.*

The mill was built in the most substantial and durable manner, timbers for the frame-work of any required dimensions being readily obtainable and cheap. Power was derived from a 150 horse-power Harris-Corliss engine and three 60 horse-power boilers, and the plant was complete in the three departments of crushing, amalgamating and concentration. As at first arranged, the crushing appliances of the system consisted of one 20 by 10 Blake breaker, the product of that going to a 30 by 5, product of the 30 by 5 going to two 60 by 2 multiple-jaw crushers, each with three jaws, receiving

* For account of Haile gold-mine and ores, see papers of E. Gybbon Spilsbury, M. E., *Transactions*, vol. xii, pp. 99-106. Ibid., vol. xv, pp. 767-770.

capacity of 20 by 2 inches. The product of the 60 by 2 multiple crushers, approximately of corn-grain size, was elevated and screened through one-quarter inch holes. That which did not pass the holes of the screens went to finer crushers. The material, ¼ inch fine and finer, was elevated, screened through 40-mesh wire screens, and all that did not pass 40-mesh, that is, material between ¼ inch and 40-mesh, went to two pairs of Krom's new swinging-block rolls, each 30 inches in diameter and 16 inches face, main driving pulleys 8 feet in diameter, 14 inches face. The product of the rolls was discharged into No. 2 elevator, elevated and screened; that which would not pass 40-mesh returning to them, together with the new supplies of coarse material from the fine crushers. The finer crushed product, 40 fine and under, was discharged from the spouts of hoppers beneath the fine screens, where it encountered a stream of water, mixing it and conveying to 10 Atwood amalgamators in two groups of 5 each. The "pulp" from each group of amalgamators, discharged at their lower ends, was carried by troughs to 20 Embrey tables, two groups of 10 each, arranged in pairs, each pair being fed from a spitzkasten on the line of the troughs conveying the pulp.

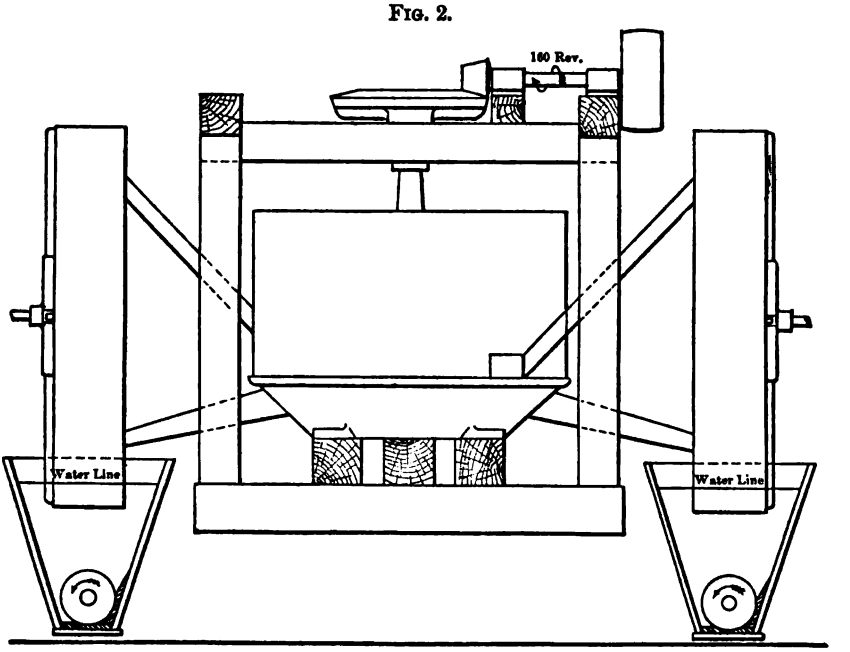
The final discharge of superfluous water and finest slimes was opposite to the central point of the concentrating room, where the flow of tailings from the two groups of Embrey tables (10 in each) was united, and finally discharged into the waste weir.

It is evident that from the 30 by 5 crusher on, the rest of the mill was arranged in two symmetrical halves, either of which, in case of necessity, could be run independently of the other. As so arranged, about 1000 tons were put through the mill, when it became evident that a change was imperative. The Krom rolls, one pair on each side of the mill, were the chief cause of frequent stoppages, often becoming surcharged and coming suddenly to a full stop, resulting either in throwing the belts or their slipping upon the pulleys, running at about 100 revolutions per minute.

Although the greatest care was taken to insure their being properly fed, this surcharging would happen. The shells or tires had in crushing 500 tons each worn down one-quarter inch; that is, one-half in the entire diameter. The surfaces had become more or less pitted and corrugated. The returns to them were increasing rapidly, and the cost of the wear of the wire cloth had been at least $1 per ton. It became evident that for purposes of *fine* crushing their use must be abandoned.

Accidental knowledge of the results obtained by many years' use of a Chili mill at glass-sand works, near Pittsburgh, Pa., led to the immediate substitution of two Chili mills (see Figs. 2 and 3) for the two pairs of Krom rolls, and running the material after being crushed to one-quarter inch *wet* instead of *dry*. This proved at once to be a solution of the question of crushing and the difficulties of fine screening.

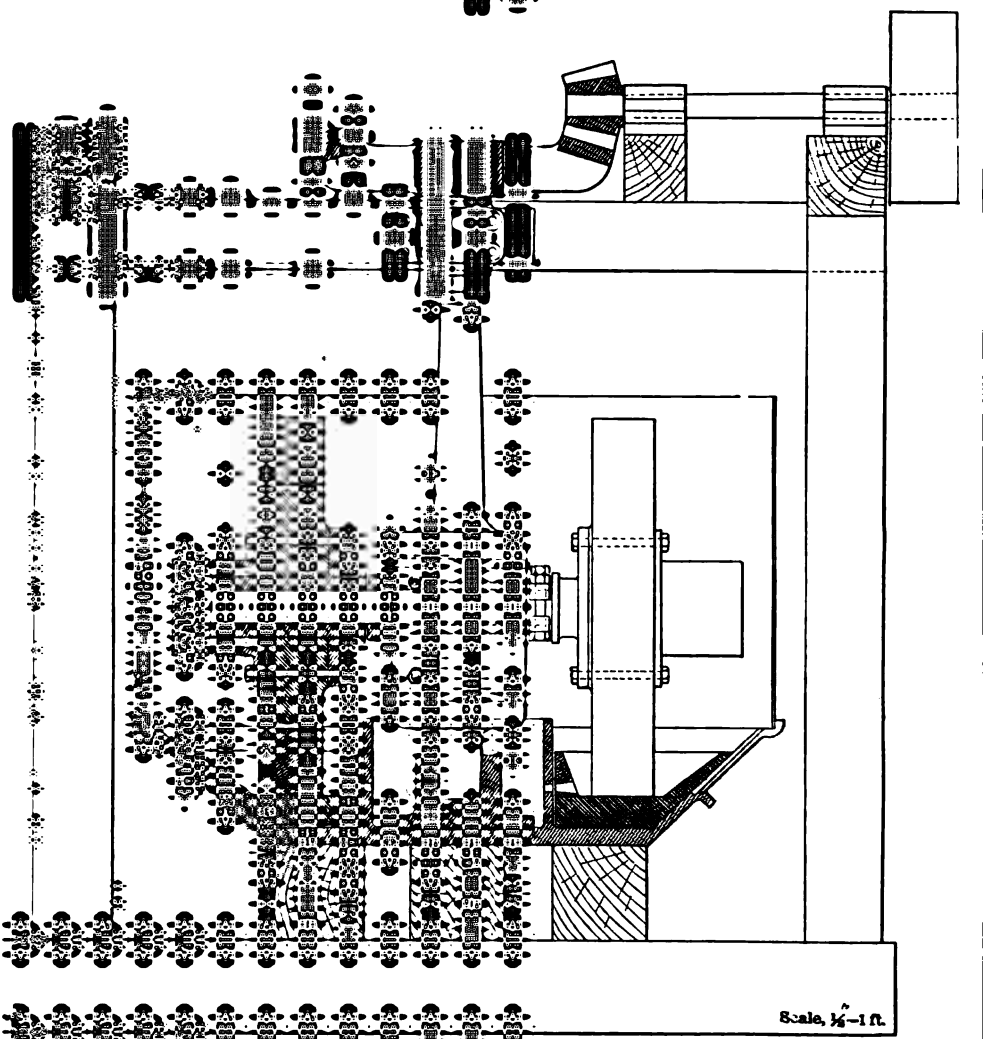
FIG. 2.



Chili Mill, with Screens outside.

The capacity of these mills, run as they were, was something wonderful,—no one who has not seen one run as it should be can form an idea of the rapidity, economy and certainty of their work, when dealing with material already carried to one-quarter of an inch and finer. The central spindle, carrying horizontal axis on which the wheels revolved, had a speed of 40 revolutions per minute. Each wheel of the mill, weighing about a ton, was 4 feet in diameter and 8 inches face. The distance from outside to outside of the wheels was 50 inches, and the tires were of hard white iron, having a cross-section 8 by 8 inches. The segmental dies on which the wheels ran were of best chilled iron. The shallow pan holding the dies was surrounded by an enclosure of sheet iron 4 feet high, to prevent water and ore being thrown out by the splash of the rapidly

...ch Chili mill was a revolving ...nd 18 inches face, turning on ...e side furthest from the mill.



...tion.

...red with wire cloth 35 meshes ...sible steel wire, equivalent to ...re of ordinary fineness. The

inner periphery of this tub-shaped screen, provided with a rim about 4 inches deep, was divided into six segments by wooden pieces across its face, forming so many buckets which served to elevate the material which failed to pass the screens, and throw it on to an apron which carried it back to the mill. This screen was run in water, submerged to the depth of the rim,—*i. e.*, about 4 inches,—and at the rate of 12 revolutions per minute.

No screens were used in the Chili mill itself; the crushed ore issued from openings on opposite sides and was, by the aid of water, discharged on to the inner periphery of revolving screens.

The substitution of Chili mills for the rolls, and the adoption of the wet method, had solved not only the problem of crushing, but that of screening. The wear of the wire cloth was at once reduced to about 10 cents per ton, and the two Chili mills proved their capacity to readily handle 7½ tons per hour of the hardest and toughest quartzite from the Blauvelt mine—on the softer and in some cases decomposed ores, as high as 20 tons per hour were put through them; in fact, the crusher could not supply the amount of softer ores they were enabled to handle.

As so arranged, the mill was run, crushing about 3000 more tons of ore. Very great improvements in economy of screening could be made, but all alterations had been made under the greatest pressure, and within the shortest possible limit of time. Material one-quarter of an inch coarse should not be thrown on to wire cloth 35 to 40 fine, especially when containing a large percentage of heavy sulphurets, as was the case in this instance, but should first go to a hydraulic separator or coarser screen. Only the approximately sufficiently fine should go to the screens, the remainder being carried back to the mill without touching the wire cloth.

For purposes of amalgamation, where concentration is not desired, it is evident that the use of the Richards-Coggin hydraulic separator would enable one to dispense with screening entirely. As to the power required, the mill showed the greatest possible economy. It was often run with capacity of 7½ tons per hour, with boilers carrying only 40 pounds of steam; engine with 80 pounds of steam rating at 150 horse-power, and I think with consumption of about 4 cords of wood per diem.

The following is a statement of percentages of different fineness of the finished pulp from the Chili mill, from sample taken and tested by Mr. Raymond, assayer of the Company:

	Per cent.
Amount passing 100-mesh screen, ...	78.98
" on 100 and through 90, ...	1.66
" " 90 " 80, ...	1.66
" " 80 " 70, ...	.....
" " 70 " 60, ...	3.32
" " 60 " 50, ...	1.09
" " 50 " 40, ...	5.01
" " 40, ...	8.30
	100.02

It is to be hoped that Mr. E. Gybbon Spilsbury, then general manager and now consulting engineer of the Haile Gold-Mining Company, will in time give in detail many points and facts here necessarily omitted. I am without special items of cost of milling per ton of ore in the Blake mill at the Haile mine, and can only state in a very general way that it showed great economy of power and material after the introduction of the Chili mills. Its operations were not sufficiently great to determine the real facts with accuracy. Enough was shown by its working to prove that mills on this system for crushing and amalgamating or concentrating, or both, may in future come into general use, that their economy per ton of ore crushed, of power and material, is greater than that of the stamp mill, and that there is no reason why they should not run with quite as much if not with more certainty and as continuously as the best stamp mill ever built.

In the summer of 1886 I furnished plans and machinery for a second mill on the Blake system to the Chateaugay Ore and Iron Co., Lyon Mt., N. Y. This was a complete crushing and concentrating plant, the crushing being done entirely by Blake crushers, and with a daily capacity of 600 long tons, from 15 inches down to $\frac{1}{4}$ of an inch, and it was built in accordance with plans shown in Figs. 4, 5, and 6. The iron-ores of this company are well known as being among the best, if not the best, Bessemer ores in the country. The deposit or vein of magnetic iron-ore is enormous, and the output from this mine often exceeds 1500 tons daily. The richer portions are sorted out and shipped, the leaner ores, consisting of magnetic iron-ore in grains, disseminated through a tough quartzose and feldspathic gangue, are sent to the mills for concentration. In order to effect this by jigging, with best results, it must be reduced to a size to pass a $\frac{1}{4}$-inch or $\frac{5}{16}$ths inch round hole. The jigs used are those known as the Conkling, with an annular revolving sieve and central discharge, said to be the in-

vention of Mr. Hooper, of the American Graphite Co., Ticonderoga, and improved in some details by Mr. W. B. Hodgson, Superintendent of Separators for the Chateaugay Ore and Iron Co. The concentrates pass into a hutch from whence they are taken out by belt elevators. The capacity of one of the jigs, as run at Chateaugay, is 100 tons of crude ore in twenty hours, or 5 tons per hour. The elevation and plans of this mill, in accompanying plates, show very clearly its construction. The ore is brought to the mill by rail, in side dumping cars, carrying on the average 7½ to 8 long tons each from the various dumps one-quarter to one-half of a mile distant. It consists of two groups or systems of crushers, with elevating and screening appliances, each group being an exact duplicate of the other; a "jack-pulley" on main-line shafting being placed centrally between them. Power is derived from a 250 horse-power Harris-Corliss engine, and battery of three 100 horse-power boilers of Parks Bros. best steel. Each group consists of the following crushers, all of Challenge pattern, one 20 x 15, crushing from 15 inches to 2 or 2½ inches.

Product of each 20 by 15 is divided, going to two 30 by 5s.

Product of 30 by 5s is elevated and screened; that passing a ¼ inch round hole is finished product, as far as crushing is concerned, and is carried to the jigs. The "coarse," 1½ to ¼ inch, goes to three 60 by 2 Multiple Crushers, each with three jaw openings 20 by 2 inches. Product of these is elevated in No. 2 elevator and screened, formerly through holes ¼ of an inch in diameter, now through $\frac{5}{16}$ths holes. That passing through the $\frac{5}{16}$ths screen-holes goes to jigs; that going through $\frac{11}{16}$ths inch holes goes to two 15 by ½ fine-crushers. Each of the 15 by ½ crushers has 7 jaw openings, with 7 openings each 15 by ½ inch. Material not passing $\frac{11}{16}$ths-inch holes, but going out the end of screen, goes back to the 60 by 2 Multiple Crushers. Product 15 by ½ fine-crushers is elevated and screened, material not passing $\frac{5}{16}$ths-holes returning to them. Each group of crushers has three Conkling jigs.

The mill was completed and started September the 26th, 1886, and was run in a desultory sort of way until the organization of day- and night-shifts, October 18th following, when its operations may be said to have commenced. Its normal capacity on ore reasonably dry was shown to be 30 long tons per hour. This was the average amount crushed hourly up to the 8th of November; then a heavy snow-storm came and difficulties due to the presence of fine wet ore interfering with proper screening were encountered. Grizzlies were

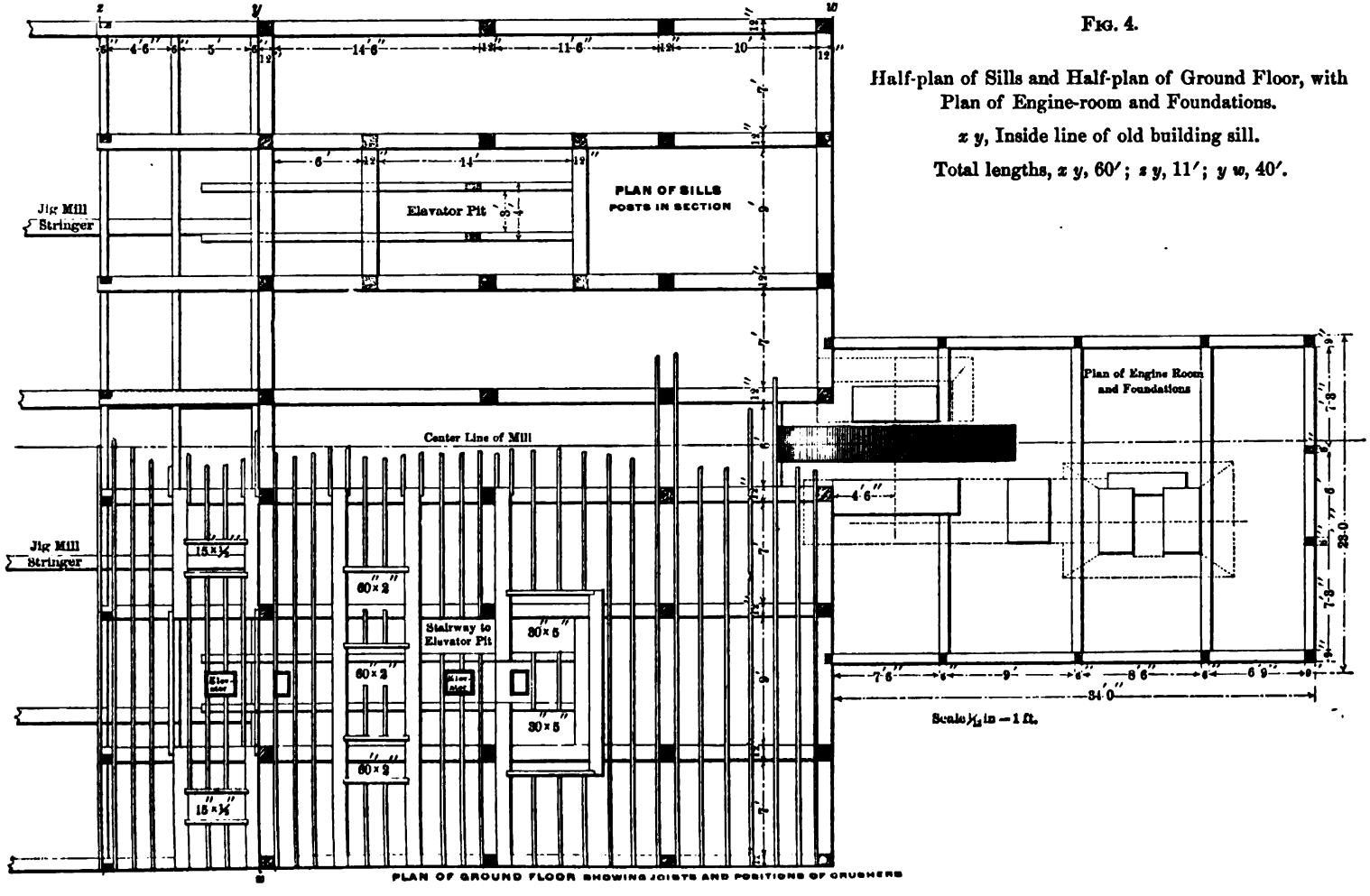


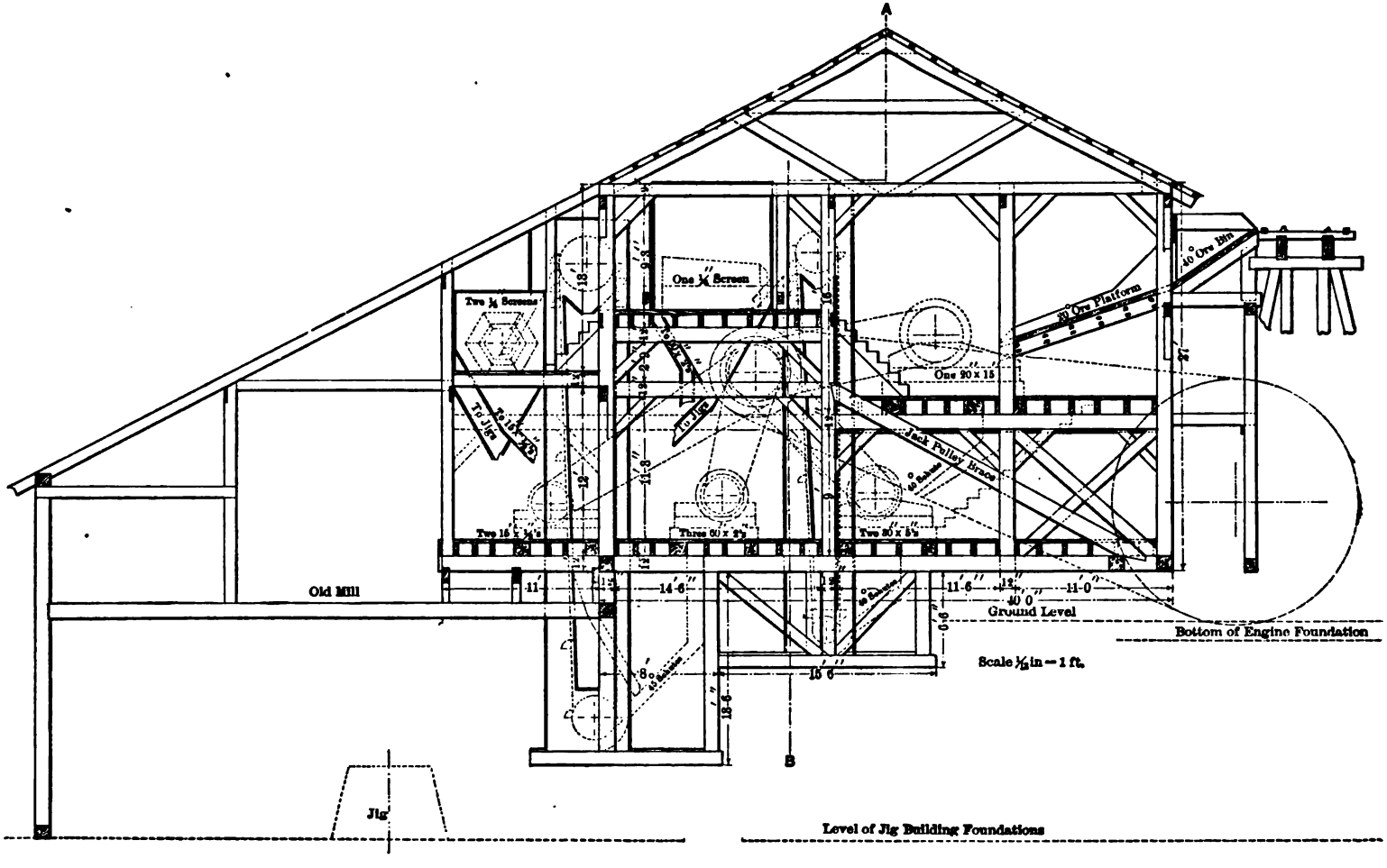
FIG. 4.

Half-plan of Sills and Half-plan of Ground Floor, with Plan of Engine-room and Foundations.

x y, Inside line of old building sill.

Total lengths, *x y*, 60′; *z y*, 11′; *y w*, 40′.

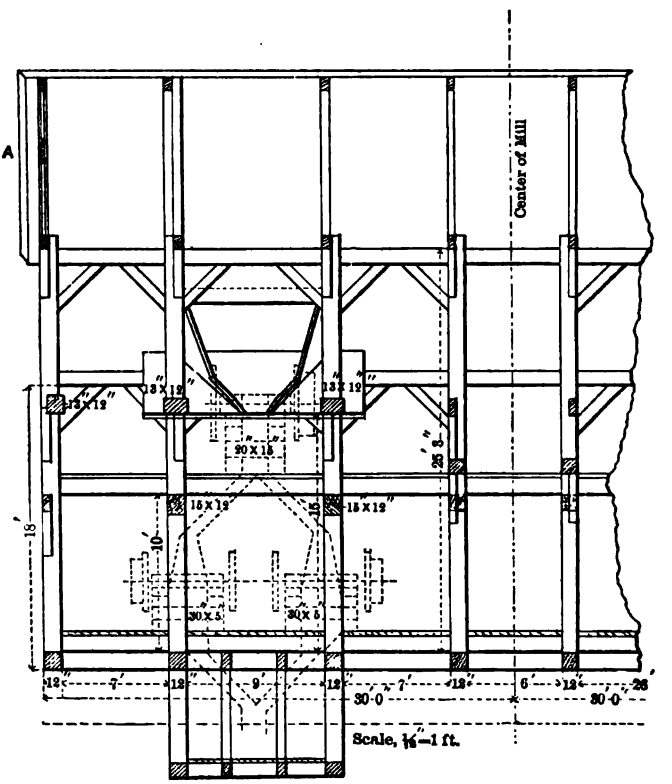
FIG. 5.



Section on Center Line of Mill (Fig. 6).

put in at the 20 by 15 crushers, the idea being to take out the fine wet ore and send it to the wet screens below. This was fairly successful, but soon, owing to the rigorous climate of that latitude, bringing heavy snows and severe cold, the fine wet ore would freeze in solid mass on the dumps or in the cars bringing it to the mill, and would fail to pass the grizzlies, but go to the crushers. It would pass the

FIG. 6.



Section on Line A B (Fig. 5).

first two crushers well enough, giving no trouble, but by the time it reached the 60 by 2 multiple crushers the heat evolved in the crushing would thaw out the cementing ice, giving damp product which would not screen readily. In this way, or by imperfect screening, the actual product of the mill was greatly reduced, and notwithstanding the impossibility of sending the ore to the mill dry, or the absence of a sufficient supply of water to run the mill wet, and in that way obviate the difficulties of screening fine or wet and

frozen ore, the mill has since that day been running uninterruptedly and continuously, day and night, to this time.

The actual amount crushed from September 26th, 1886, to January 1st, 1888, being 122,814 long tons or 137,551 short tons, at a cost for crushing and concentration of $42,200.55, or 34.36 cents per long ton or 30.67 cents per short ton, distributed as follows:

2018$\frac{1}{2}\frac{1}{4}\frac{1}{4}\frac{1}{0}$ tons of coal,	$8,612 37
Labor,	21,008 72
Oil waste, etc.,	2,374 30
Mill supplies, renewals, and repairs,	10,205 16
Total,	$42,200 55

DISTRIBUTION PER TON OF ORE.

	2240 lbs.	2200 lbs.
Fuel,	7 cts.	$6\frac{1}{4}$ cts.
Labor,	17 "	$15\frac{1}{4}$ "
Oil waste, etc.,	2 "	$1\frac{7}{16}$ "
Mill supplies, renewals, and repairs,	8 "	$7\frac{1}{2}$ "
Total,	34 "	$30\frac{7}{16}$ "

This economy is certainly remarkable, and still more so when we consider the prevailing unfavorable conditions as regards successful screening of the ore. Had the ore been reasonably dry instead of being generally wet and, during the winter months, frozen, or if the crushing after the passage of the 30 by 5 crushers had been "wet" instead of "dry" and the screening in that way made perfect as it can be, the actual, average, daily product could have been increased, even doubled, and the cost of crushing and concentrating per ton of crude ore reduced to less than twenty-five cents. In a recent number of the *Engineering and Mining Journal*, January 28th, 1888, giving a description of the Tamarack steam-stamp, built by Messrs. E. P. Allis & Co., of Milwaukee, the capacity is given at 225 tons per day of 24 hours, on the basis of 34 to 36 tons of ore for every ton of coal consumed. In the Chateaugay mill the total consumption of coal in crushing from 15 inches down to $\frac{1}{4}$ inch, as opposed to 3 inches down to $\frac{5}{16}$ths of an inch at Lake Superior, as proved by the results already stated, is one ton of coal to 68 tons of ore. If allowance were made for the coal used at Chateaugay mill in heating the mill itself and the water used in concentration, it is perfectly safe to assert that the Ball stamp, in its highest condition of development and efficiency will give crushed product in tons not more than half

as great per ton of fuel burned as a mill constructed and run upon the Blake system. But the consumption of fuel in any method of stamping is, as a rule, a matter of small consequence compared to the losses due to the production of slimes not susceptible of concentration. This is, and must always be, an insuperable objection to their use where subsequent concentration or lixiviation are proposed.

If it were possible, and I believe it is, to so construct a crushing-plant consisting of crushers alone as to handle the copper-ores of Lake Superior, crushing first down to ½ inch or larger, then jigging, recrushing, and jigging again, and so on, it would certainly be a great improvement on the very simple but extravagant methods at present pursued. Certainly a large percentage of the loss there is due to the abrasion of copper beneath the stamp, necessarily involved in carrying all the material to $\frac{3}{16}$ths of an inch before it can escape from the battery. Only costly experiment can determine this, but there is every reason to believe that multiple-jaw crushers can be built of such design and strength as to enable them to flatten out and pass masses of "included copper" which may by accident escape the attention of the attendant and go to them. Ordinary Blake crushers are and have been in general use at Lake Superior for many years in preparing the rock for the stamp; that is, in breaking it down to 2½ to 3 inches, pieces of metallic copper that will not pass the lower jaw opening being picked out by hand before going to them.

In the operation of multiple-jaw crushers as already built, the intrusion of foreign material, such as cast iron, bits of steel, pick, gad and drill points, is a frequent occurrence. They are either crushed or simply render the jaw in which they are found inoperative until they are removed, which is very easily done. No breakage of the machine has ever been known to occur from such causes. In a fine crusher I have known a ½ inch steel set screw to remain for a day and to be actually drawn down to a perfect wedge-shaped mass, without any injury to the machine. The packing of all the jaws with damp, fine ore intermingled with the coarse is a far more serious matter, and must be avoided by proper screening.

Experience has fully demonstrated this remarkable fact, that the Blake multiple crusher will do its work with as much ease and certainty and with greater *absolute* economy for wear and tear per ton of ore crushed, than an ordinary Blake breaker. The explanation of this lies first, in the multiplicity of jaws affording a safety

provision, and secondly, in the economy of the use of the finest quality of hardened tool-steel for wearing-surfaces.

Whether the quartz-mill of the future will consist of a system of breakers supplemented by Chili mills or other fine crushing devices, remains to be seen. Certainly, if we combine the results of crushing at the Chateaugay mill down to ½ inch and at the Haile mill from ½ inch down to 40 fine by means of a Chili mill, we show the greatest possible economy in crushing. The wear on the Chili mills used at the Haile mine, in crushing 3000 tons of ore was, on the diameter of the wheels, ¼ inch; on the segmental dies, ¼ to ⅜ of an inch. The total wear of iron in these mills per ton of ore is easily calculated. On the four wheels of the two Chili mills it would be 156.64 pounds, and on the two sets of segmental dies it would be 205 pounds, making a total of 361.64 pounds, equal, on 3000 tons, to .12 of a pound of iron to a ton of ore.

It would seem probable that multiple-jaw crushers could be used with advantage in supplementing the ordinary crusher for preliminary crushing before feeding stamps. This is only true to a certain extent.* But if the crushing be carried beyond a certain limit or down to coarse sand, for example, the practical result would be to reduce the product of the stamp mill rather than to increase its capacity. The reason of it is that ore so crushed, when fed to the stamps would simply bank up or pack beneath them. It requires coarse with the fine to "root up" the fine under the stamp and prevent its packing. It is probable, also, that if material were confined in a Chili mill by screens surrounding it instead of being sent to screens entirely outside of the mill, as was the case at Haile, it would pack beneath the wheels and the results have fallen short of those obtained.

The Blake crusher, as a machine, has an extended and enviable reputation for strength, simplicity, and durability. Their use in a "system" forming complete crushing plants has already led to many improvements and perfection in their details. Their combination in a system so as to run continuously and without interruption is indeed a matter demanding no inconsiderable skill, care, and experience. But the results at Chateaugay and at the Haile mine have been such as to completely demonstrate the economy of this method of crushing, and it seems to me that these results possess more than ordinary interest for the engineer.

* See "Notes on Gold-Mill Construction," by Augustus J. Bowie, Jr., *Transactions*, x., 98, a paper read at the Virginia Meeting, May, 1881.

METHOD OF CONSTRUCTING STRATA-MAPS TO REPRESENT STRATIFICATION OR BEDDING.

BY JAMES T. B. IVES, F.G.S.

(Communicated by request of the Council at the Boston Meeting, February, 1888.)

THE map exhibited* as an example of my method of constructing geological strata-maps is essentially an educational appliance. The method, however, is available for the production of maps of comparatively small areas, which, I am confident, will be found of great practical value to mining engineers. Before stating how the method may be so applied I will briefly describe the map exhibited, as the easiest way of making my meaning clear.

Almost any region might be taken. This map represents the eastern half of the United States; and the geology is that of the map issued by the Geological Survey, and revised and extended subsequently by Professor Hitchcock for your Institute. I use ten cards, of the colors used in the geological map, each having the geography of the area printed upon it; and I superimpose these in the order in which the colors occur in the index to the map. I next cut away a portion of each card, excepting the lowest, leaving only so much of each as represents the exposure of the formation of that color plus all those which overlie it. When that is done, the whole, looked at from a distance, appears to be an ordinary geological map. Looked at more closely, however, the formations are seen to overlie one another, as in nature. The cards are covered with glass in a frame, which, hinged at the bottom edge, can be let down. The cards are then found to be all hinged in the same manner, and they, too, may be lowered, one after the other.

I am told by teachers of geology of great experience and the highest reputation that by this appliance they will be assisted in imparting to students the first facts and principles of geological science. Having let down the whole series of cards, until that representing the Archæan is seen underlying all the rest, the teacher will gradually rebuild it. The Cambrian will thus be *laid down* upon the Archæan before the eyes of the class, the Silurian upon that,

* A map of part of the United States, constructed on the basis of the map in Volume XV. of the *Transactions*, was exhibited in illustration of this communication.

then the Devonian, Carboniferous, Cretaceous, and so on. The teacher will explain that the superposition of the cards corresponds with that of the strata in nature; that the cutting-away of portions of the cards corresponds to the work of erosion or denudation in nature; and that the steps formed by the thickness of the cards correspond to the escarpments of nature. He will show how the underlying cards crop out from beneath those by which they are overlaid, corresponding to natural outcrops. He will tilt the map, and illustrate dip and strike in nature. He may bend the cards so as to produce a synclinal trough or an anticlinal ridge. Inliers are beautifully exhibited where holes are made in higher cards to show lower ones. Outliers have to be produced by attaching a portion of a higher card, in place, to that underlying it. Conformability in stratification is also illustrated in the map.

In the application of this method to the requirements of the mining engineer, the area to be represented should be small, since the scale needs to be large. The layers of the map may then correspond to single formations instead of systems, as in the map exhibited, or they may correspond merely to beds, all belonging to the same formation. Nor need they be all of the same thickness throughout and spread out horizontally. Instead of cardboard, *papier maché* may be used, shaped and made of variable thickness, to correspond with the several beds to be represented. On the successive layers of such a map notes may be made of ascertained thickness, mineral characters, or underground workings, as the data are obtained.

So much for the strata-map. Arising out of it I offer two suggestions:

1. Instead of corresponding to natural beds of distinct geological age the cards may correspond to units of vertical measurement (say 10 feet or 100 feet each), still using colors, but graduated. On these such records as already suggested may be made from time to time.

2. This method is available for contour-maps, each card representing a certain unit of elevation (as 10 feet or 100 feet). Starting, as in any contour-map, from a given datum-line or base, superimpose so many of the cards as may be necessary, and so shaped as to produce the contour-effect required. Various colors, graduated or otherwise, may be used. In many cases a series of browns, using the darkest at bottom and lightest at top, would be advantageous, producing a perspective effect.

In conclusion, I take this first opportunity of stating that,

although I made my first strata-map (of England and Wales) for my own use eighteen years ago, and have never heard or seen anything at all like it until lately, it has come to my knowledge within the last few weeks that Prof. Charles A. White, in his Geology of the State of Iowa published in 1870, used the same method for illustrating the geology of that State. That author's purpose appears to be purely to exhibit the underground extension of the several formations; whereas my purpose is emphatically to illustrate the phenomena of the science generally.

THE NEW YORK MINING LAW.

BY R. W. RAYMOND, NEW YORK CITY.

(Boston Meeting, February, 1888.)

THE Mining Law of New York, contained in Title XI., Chapter IX., Part One of the Revised Statutes, is as follows:

SECTION 1. The following mines are, and shall be, the property of the people of this State in their right of sovereignty:

(1.) All mines of gold and silver discovered, or hereafter to be discovered, within this State.

(2.) All mines of other metals discovered, or hereafter to be discovered, upon any lands owned by persons not being citizens of any of the United States.

(3.) All mines of other metals discovered, or hereafter to be discovered, upon lands owned by a citizen of any of the United States, the ore of which, upon an average, shall contain less than two equal third parts in value of copper, tin, iron, and lead, or any of those metals.

SEC. 2. All mines and all minerals and fossils discovered, or hereafter to be discovered, upon any lands belonging to the people of this State, are, and shall be, the property of the people, subject to the provisions hereinafter made to encourage the discovery thereof.

SEC. 3. All mines of whatever description, other than mines of gold and silver, discovered, or hereafter to be discovered, upon any lands owned by a citizen of any of the United States, the ore of which, upon an average, shall contain two equal third parts or more, in value, of copper, tin, iron, and lead, or any of those metals, shall belong to the owner of such land.

SEC. 4. Every person who shall make a discovery of any mine of gold or silver within this State, and the executors, administrators, or assigns of such persons, shall be exempted from paying to the people of this State any part of the ore, produce or profit of such mine for the term of twenty-one years, to be computed from the time of giving notice of such discovery, in the manner hereinafter directed.

SEC. 5. No person discovering a mine of gold or silver within this State shall work the same until he gives notice thereof, by information in writing to the Secre-

tary of this State, describing particularly therein the nature and situation of the mine. Such notice shall be registered in a book to be kept by the Secretary for that purpose.

SEC. 6. After the expiration of the term above specified the discoverer of the mine, or his representatives, shall be preferred in any contract for the working of such mine, made with the legislature or under its authority.

SEC. 7. Nothing contained in this title shall affect any grants heretofore made by the legislature to persons having discovered mines, nor be construed to give any person a right to enter on or break up the lands of any other person, or of the people of this State, or to work any mine in such lands, unless the consent, in writing, of the owner thereof, or of the commissioners of the land office, when the lands belong to the people of this State, shall be previously obtained.

SEC. 8. In all cases in which a person or persons shall have discovered a mine or mines, and become entitled to work the same, pursuant to title eleven, chapter nine, part first of the Revised Statutes, and such person or persons shall form a corporation pursuant to chapter forty of the laws of eighteen hundred and forty-eight and the several acts subsequent thereto and amendatory thereof, if the consent in writing to enter upon and break up the lands of any person in or upon whose lands said mine or mines are found, shall be refused or cannot be obtained by agreement, or by reason of the infancy or the absence of such person from the State, or other legal disability of the owner of such lands, the said corporation so formed may enter upon and break up the lands of such person for the purpose of working such mine or mines in the manner hereinafter provided, and the right and easement so to do shall be deemed granted for public use, and for the public purpose of obtaining minerals reserved to the State, and the said right and easement are hereby granted to the corporation so formed, on their filing with the commissioners of the land office a full description of the location of such land, and obtaining a grant therefor from said commissioners, who are hereby authorized to make such grant and file the terms thereof.

SEC. 9. The said company entitled to work such mines may file a petition in the Supreme Court of the State, setting forth the facts upon which they claim such right, and the reasons which prevent their entering upon the land necessary for their mining operations; and upon such petition the court may appoint three disinterested persons as commissioners to examine into the matter, ascertain and affix the damages aforesaid, and report to the court. Notice of the filing of such petition shall be published in one of the papers printed in the county, or in each of the counties where the mine or mines are situated, and in the State paper, and a copy of such notice shall be served personally upon the owners of the land; or, if they are infants, upon their guardians; or, if lunatics, or under any other legal disability, on the committee having charge of them and their property. And the publication of such notice in the State paper shall be deemed a sufficient notice to such owners as are residents in other States or in other countries, or are temporarily absent from the State, provided that when the actual residence of such absentee is known or can be ascertained, a copy of such notice and petition shall be sent them by mail. All the parties interested shall be entitled to a hearing before such commissioners, at such time or times as said commissioners shall appoint. The report of the commissioners shall state:

(1.) The existence of the mine or mines proposed to be worked:

(2.) The names of the parties owning the land in which the mine or mines are situated, and the owners of the adjacent lands, so far as they are affected by the application, and the nature and value of their interest in the same, individually. A

map of such lands, from actual survey by metes and bounds, shall accompany the report.

(3.) An estimate of the damages to such owners from the contemplated use and occupation of their lands.

(4.) Such other information as the court may direct.

SEC. 10. The report of the commissioners shall be made within a reasonable time, to be fixed by the court. An order shall be made, in the discretion of the court, either denying the petition or granting it, determining the quantity of land necessary for working the mine or mines, the damage to the property by taking possession thereof, and the annual rent or the compensation to be paid to the owner, lessee, or occupant thereof, so long as the use and occupation shall continue. And thereupon the company in whose favor the order shall be made, upon payment of the damages and upon entering into an agreement, to be approved by the court, to pay the annual rent of the compensation and damages thus determined, shall have the right to enter upon and occupy and use the land set apart by such order, or so long as they or their assignees shall work the said mine or mines, and shall pay the said annual rent or compensation.

SEC. 11. If the parties owning the land are infants, or otherwise incompetent to act, the court shall appoint guardians to take care of their interests, and shall direct how any damages assessed, or compensation or rents to become due, shall be paid and invested to their benefit.

Some months ago a foreign consular officer, engaged, under instructions from his government, in collecting the mining statutes of the various States, applied to me for an explanation of the peculiar provisions of this law. At that time I was unable, by reason of multiplied other labors, to bestow on the subject any attention; and I was obliged to confess frankly my ignorance of the meaning and history of this strange piece of legislation. I promised my visitor (and myself), however, that I would examine the question at the first available opportunity; and I now present this paper, not so much because I have been able to pursue the matter to a satisfactory conclusion, as because I have come to a gap which I have been thus far unable to cross, but which may, perhaps, be bridged by the labors of others, more favorably situated as to the apparatus and the leisure for investigation.

Meaning of the Law.—The word "mine" is evidently used throughout in the sense of a mineral deposit upon which a mine may be opened. The distinction between mines and quarries or diggings, found, for instance, in the French code, is not recognized here.

The first three sections apparently assume the sovereign right of the State to all mines of whatever metals or metallic ores within its borders, and proceed to divide this property of the State into two classes, one of which is bestowed upon the private land-owner, and

the other reserved to the State.. In the first class are put practically copper-, tin-, iron-, and lead-mines only, and not even these, unless the land-owner be a citizen of the United States. Into the other class go all the rest.. The peculiar phraseology of the third article of Section 1, and its converse in Section 3, seems to be intended to protect the citizen land-owner possessing a mine, the chief value of which is in one or more of the four metals named, from any claim of the State based on the presence of some other metal in subordinate value.

No other than metallic minerals are provided for, except in Section 2, where the terms "all minerals and fossils," covering coal, petroleum, building-stone, cement, lime, soapstone, clay, corundum, phosphates, mica, saltpeter, graphite, asphaltum, salt, gems, etc., seem to be, as, indeed, the whole section seems to be, mere surplusage. For, at first consideration, one is inclined to say that the ownership of the State in the minerals on its own lands could not be questioned. Since there is no other sovereign to dispute it, why need it be re-enacted? But this section may have been intended to assert the rights of the State of New York against any possible claim of the Federal government. Or it may have been intended to bar any assertion, on the part of prospectors on the State lands, of a right independent of the statute, arising out of the discovery itself—like the ancient *Bergrecht* of the Germans, which was rather recognized than created by the sovereign.

By implication, then, every non-metallic mineral deposit in the State is the property of the land-owner, whether he be a citizen or not.

Gold- and silver-mines belong to the State without restriction. Even if the ore of a mine be principally valuable for copper, tin, lead or iron, there is no grant of it to the land-owner, if the remainder-value be gold or silver. A mine of argentiferous galena, or auriferous copper pyrites, for instance, belongs to the State, however small the proportion of the precious metals. It is to be presumed that in construing this provision, judges and juries would require, to constitute a mine of gold or silver, a sufficient proportion of the precious metal to be actually a source of income to the miner. But further than this, I do not see how they could go. The only remedy for any hardship involved would be new legislation.

Sections 1 and 3 practically assert State ownership of all mines of other metals than gold or silver, even on the lands of citizens, and, on the other hand, concede to the land-owner, that if the mines

in question do not contain of other non-precious metals than copper, tin, lead, and iron, on the average, at least one-third of their total value, then the State will not claim them. To put it in homelier phrase, it is as if the State had said "Mines of copper, etc., belong to the citizen land-owner; mines of all other non-precious metals, to the State. But if two-thirds of the value of an ore is copper, etc., then the State will consider it as a copper, etc., mine, whatever may be the other non-precious metals contained in it."

The principal substances included in the phrase "all other metals," are manganese, zinc, quicksilver, nickel, cobalt, platinum, and iridium. It is a curious fact that, so far as I know, there are not now working in the State any mines carrying these metals, in which the two-thirds value of copper, iron and lead is not present. The old mine of niccoliferous pyrrhotite, with scattered bunches of chalcopyrite, at Anthony's Nose, on the Hudson, was, perhaps, at one time, an exception; yet I have never heard that the State had asserted its claim to that. Zinc-mines falling under this provision might easily occur; as, for instance, if the red zinc-ore of New Jersey should ever be found in the State. The same is true of manganese deposits.

The mention of tin with copper, lead and iron has had no practical result, since no tin-mines have been found in New York. Nor have manganese deposits been opened in the State, so far as I can recall.

Sections 4, 5, etc., which regulate the working of mines belonging to the State, are curiously defective. Section 4 rewards the discoverer of gold- or silver-mines (whether on his own land, or the land of another, or the land of the State) with an exemption from royalty for twenty-one years. Section 5 requires the filing of a notice by the discoverer, and implies that, after such notice, he may proceed to work the mine as his own for twenty-one years; and Section 6 promises to him or his representatives, at the expiration of that period, the preference in any contract made with the State authority for further working. No proof of the discovery is required; and there is no requirement of a notice from any person, *not the discoverer*, proposing to work a gold- or silver-mine. In Section 7, entering, prospecting and mining on any land, without the previous written consent of the owner, is forbidden; or, to speak more precisely, it is declared that nothing in *this* law shall be construed to authorize such trespass. Apparently, therefore, no person who had discovered a mine without the previous formal consent of the land-owner, could take the benefit of this law in any way. Yet the

land-owner could not come forward and represent himself to be the discoverer, so as to secure the twenty-one years' exemption from royalty. On the other hand, not being the discoverer, he would not be obliged, under Section 5, to give notice. For all that I can see, he might work the mine to his heart's content, because he would be merely digging beneath the surface of his own land; but, if he undertook to dispose of the product, the State could come down upon him under Section 1.

But Sections 4, 5 and 6 refer to gold- and silver-mines only. For all the other mines, declared by the third article of Section 1 to belong to the State, there is no provision made, under which the right to work them may be required. Whether the land commissioners could make regulations on the subject, under the general authority of the law, is questionable. At all events, they have not done so. A recent case in my own practice illustrates this point. I was consulted in behalf of a citizen of the State, who had discovered on his own land what he believed to be a deposit of platinum, and asserted that the character and value of the material had been determined by official assay at the Government assay-office. What I said as to the probable value of his discovery was *obiter dictum*, and need not be here repeated. What he wanted was advice as to his legal position. Assuming that he had found a platinum-mine, of which two-thirds the value was not copper, etc., it was clearly the property of the State, and could be utilized only on terms fixed by the State. But the Commissioners of the Land Office, he said, refused to entertain any application from him, because they dealt with gold- and silver-mines only. There was nothing in the law to meet his case.

One further observation may be made as to the curious two-thirds rule embodied in Sections 1 and 3. It requires the determination of the value of the ore "on an average" as a prerequisite to the determination of the title to a discovered mine. This is the most impracticable condition I have ever encountered in a mining law. The location of an apex is easy in comparison. How is the average value of the ore to be ascertained, before the mine has been opened? Who will pay for opening the mine, while it is yet undetermined who is to own it? The law does not even promise in such cases, as in the cases of gold- or silver-mines, that the discoverer shall have release of royalty, or any other preference. And what is "the average value" of the ore? If it be the average value of one day's or month's product, then the next day's or month's product may take the mine away from the land-owner, and give it to the State, or *vice versa*. If it be the

average of the total product from the commencement of operations, then the situation is scarcely less precarious. In either case, not only the variations in the ore, but also the fluctuations in the market, affect the ownership. Take, as an instance, a mine producing zinc-blende and galena, which are concentrated and sold. Their relative value in the ore might very likely fluctuate about the ratio 1 : 2. And whether zinc went up or lead went down, in the ore or in the market, the unfortunate mine-owner would lose his title.

But perhaps it is intended that the "average value" shall be determined once for all, and that the title shall thus be fixed at the outset. Assuming that the law as it stands would bear this construction, and that, after some fashion, a determination could be made, valid for legal, though worthless for all other purposes, it would evidently be the interest of the land-owner, after discovering the mine, to watch for a fall in the zinc-market, and then come forward with his assays, and claim the property.

Enough has been said about the absurd provisions of the law in these respects. It could never have remained on the statute-book until now, except as a dead letter in all respects but one, namely, its grant to the land-owner of iron-mines, which, at this time, covers about all the valuable working metal-mines in the State. Its reservation to the State of gold- and silver-mines and the "other metal" mines described, has never amounted to anything. A lot of notices have been filed with the Secretary of State; but I have never heard of any New York gold- or silver-mine that passed through the 21-year period, and was worth working at the end. In short, the provisions of the law that would have worked mischief, have never been called upon to work at all. Seldom has a blunder in legislation been so completely neutralized by Providence.

History of the Law.—The peculiar features of Sections 1 and 3 have been substantially re-enacted from an Act of February 6, 1789, entitled "An Act Relative to Mines" (*N. Y. Session Laws*, 12th Session, ch. xviii). This appears to have been the first mining law passed in this State. It was the result of several petitions to the legislature, as the following extracts from the Assembly Journal will show.*

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* I am indebted to Mr. Otto C. Wierum, Jr., of the Columbia College Law School, for valuable assistance in tracing the origin of this legislation, and verifying the references given.

On page 16 of that journal is found this record:

"Dec. 20, 1788.—A petition of Daniel Brinsmaid and Edmund Curtiss, of Fredericksburgh precinct, in Dutchess County, relative to a mine discovered in their own land, was read and referred to Mr. Verplanck, Mr. Winn, and Mr. Tappin."

And on page 21:

"Dec. 24, 1788.—A petition of Cornelius Newkerk and twenty-eight other persons, holding or claiming lands in Hurley, in Ulster County, relative to a mine, was read and referred to Mr. Verplanck, Mr. Winn, and Mr. Tappin."

And on page 40:

"Jan. 3, 1789.—Mr. Verplanck, from the Committee to which was referred the several petitions of Thomas Schoonmaker and others, relative to mines by them discovered, reported that from the many applications which have been made to the Legislature, and from the probability that similar applications will be frequent, it is the opinion of the Committee that it is most expedient to bring in a general bill, relative to mines. That the Committee have prepared a bill for that purpose, and have directed him to move for leave to bring in the same."

The bill was read for the first, second, and third times respectively on January 3d, January 5th, and February 2d. Upon neither of these occasions was there any debate. The passage of the bill through the Senate was equally uneventful; and it finally became a law February 6th, 1789. From the entire absence of discussion, we may fairly infer that the provisions of the bill were satisfactory to Daniel Brinsmaid, Cornelius Newkerk, Thomas Schoonmaker, and the others in Dutchess, Ulster, etc., whose petitions gave rise to it. I think we may be sure that if they had not been suited, they would have attempted, at least, to get the bill amended or defeated. Unfortunately it has been found impossible to recover the substance of those petitions. The following letter presents the final result of all researches in that direction:

STATE LIBRARY, DEPARTMENT OF MANUSCRIPTS,
ALBANY, N. Y., April 9th, 1888.

"DEAR SIR: Yours of the 7th instant has been received; and I regret to say that the petitions mentioned by you are not on record, either in the 43 volumes of Assembly Papers (1775–1830), or in the Land Papers (1643–1803).

"I believe that they were lost, *i.e.*, sold by an untrustworthy janitor, some 10 or 15 years ago, who had access to the Committee Rooms in the old capitol, where they had been stored under a resolution of the Assembly of April 20, 1830."

FERNOW (State Librarian).

English Precedents.—The report accompanying this bill from Mr. Verplanck's Committee, in 1788, indicates that no mining law had previously been passed by the Legislature of the State, since the adoption of the Constitution in 1777. Before that event, the law of the State on this subject was found in an English statute of 1688 (1 William and Mary, Chap. 30) as explained in 5 and 6 William and Mary, Chap. 6. The former of these was chiefly a repeal of an ancient law of 1404 (5 Henry IV.), regarding "the craft of multiplying gold and silver." In a brief final section only did it touch the subject of mines. This section is as follows:

"§ 4. Provided also, and be it further enacted by the authority aforesaid, that no mine of copper, tin, iron, or lead, shall hereafter be adjudged, reputed, or taken to be a royal mine, although gold or silver may be extracted out of the same."

It will be noticed that the metals here enumerated, the mines of which are to belong to the land-owner, are the same as those named in the New York law; and the occurrence in the latter list of tin, which would scarcely have suggested itself otherwise to the New York legislators, makes it probable that the English statute was in their minds. At first glance, the earlier law appears to be the more liberal, since it grants the mines of certain metals without specifying any condition as to the relation of value between those metals and other constituents of the ore. But, as I have already observed, this specification in the New York law was probably intended as a protection to the land-owner. The liberality of 1 William and Mary was too vague.

Five years later (1694), 5 and 6 William and Mary, Chap. 6, furnished a more detailed explanation of the brief general concession already made. The text is as follows:

"*An Act to Prevent Disputes and Controversies Concerning Royal Mines.*

"I. *Whereas*, by a clause in an Act of Parliament, made in the first year of their Majesties' reign, intitled 'An Act to Repeal the Statute made in the Fifth Year of King Henry IV., Against the Multiplying of Gold and Silver,' it is, amongst other things, enacted 'that no mine of tin, copper, iron, or lead shall hereafter be adjudged, reputed, or taken to be a royal mine, although gold or silver may be extracted out of the same;' but notwithstanding the good provision by the said statute to prevent the discouraging their Majesties' good subjects, who have mines of copper, tin, iron, or lead in their soils, from digging and opening the same, many doubts and questions have arisen upon the said statute, whereby great suits and troubles have arisen to many owners and proprietors of such mines; wherefore, for the better explanation of said statute:

"II. Be it enacted and declared by the King's and Queen's most excellent majesties, by and with the advice and consent of the Lords Spiritual and Temporal, and the Commons in this present Parliament assembled, and by the authority of the same, That all and every person or persons, being subjects of the Crown of England, bodies politick or corporate, that now are or hereafter shall be the owner or owners, proprietor or proprietors of any mine or mines within the kingdom of England, Dominion of Wales, or Town of Berwick upon Tweed, wherein any ore now is or hereafter shall be discovered, opened, found, or wrought, and in which there is copper, tin, iron, or lead, shall and may hold and enjoy the same mine or mines and ore, and continue in the possession thereof, and dig and work the said mine or mines, or ores, notwithstanding that such mine or mines, or ore shall be pretended or claimed to be a royal mine or royal mines, any law, usage, or custom to the contrary notwithstanding.

"III. Provided always, and be it enacted and declared, That their Majesties, their heirs and successors, and all claiming any royal mines under them, shall and may have the ore of any such mine or mines in any part of the said Kingdom of England, Dominion of Wales, or Town of Berwick upon Tweed (other than tin ore in the counties of Devon and Cornwal) paying to the proprietors or owners of said mine or mines wherein such ore is or shall be found, within thirty days after the said ore is or shall be raised and laid upon the bank of said mine or mines, and before the same shall be removed from thence, the rates following (that is to say) for all ore washt, made clean and merchantable, wherein there is copper, the rate of sixteen pounds per ton; and for all ore washt, made clean and merchantable, wherein there is tin, the rate of forty shillings per ton; and for all ore washt, made clean and merchantable, wherein there is iron, the rate of forty shillings per ton; and for all ore washt, made clean and merchantable, wherein there is lead, the rate of nine pounds per ton. And in default of payment of such respective sums as aforesaid, it shall and may be lawful for the owners and proprietors of the said mine or mines, wherein such ore is, are or shall be found, to sell and dispose of the said ore to his or their own uses; any law, statute or custom to the contrary notwithstanding.

"IV. Provided always, That nothing contained in this Act shall alter, determine or make void the charters granted to the tinners of Devon and Cornwal, by any of the Kings and Queens of this realm, or any of the liberties, privileges, or franchises of said tinners, or to alter, determine or make void the laws, customs or constitutions of the stannaries of Devon or Cornwal, or any of them."

For the origin of these concessions to the land-owner, we should have to go still further back to the celebrated "Case of Mines" (*Queen* v. *Earl of Northumberland*) in the reign of Elizabeth. The decision in that case is reported in 1 *Plowden*, 310. It is too voluminous to be given at length in this place; but the following summary, prepared by Mr. Wierum, states its main points with sufficient fulness, omitting much redundant and irrelevant matter.

The facts of this case are, briefly, as follows: The Queen sent her agents into lands of the Earl to 'dig, and win, and carry away' for her, certain copper-ore containing gold and silver; which the agents had done to the extent of 600,000 pounds weight of ore, and were continuing to do when interrupted by the Earl; whereupon the Queen brought suit for damages in £1000.

The principal claim advanced on behalf of the sovereign was of her right by pre-

rogative (1) to all mines of gold and silver; (2) to all mines of other metals containing gold or silver.

Her arguments on the first point were, in brief (a), that "common law which is founded upon reason, appropriates . . . . things most excellent to the persons who excel all others;" (b) the necessity of gold and silver as treasure for the defence of the realm and (c) for convenience sake as a medium of exchange. On the second point the chief argument was, that as the sovereign could get the gold and silver to which she was entitled, only by melting the copper, and as she was not bound to melt copper for the benefit of any one else, from the necessity of the case the gold or silver drew the baser metal with it as appurtenant (in analogy to a familiar principle of the law in some other instances).

On the first point the Earl practically conceded the general principle, but set up that the words of his grant of the lands in question from Queen Mary—"omnes et singulas mineras"—expressly conveyed the Crown's prerogative right even to gold and silver mines.

On the second point he insisted that the Queen must show that the value of the precious metal in the mixture was at least equal to that of the base, and failing such proof the ownership of the copper should determine that of the whole; "For otherwise the Crown would have all the mines of base metal in the realm, for (as I have been informed by those who are experienced in that art) there is no mine of tin, copper, lead, or iron but some gold or silver may be drawn out of it."

In the decision, the twelve judges agreed that mines of gold or silver in the lands of the subject belonged to the Crown.

All, except three, agreed that a mine of base metal containing gold or silver in the lands of a subject belonged to the Crown. The three contended that in such case the value of the precious metal must exceed that of the base (but in this case the three gave judgment on this point with their brethren, as the burden of proof was on the subject to show that the gold or silver did not exceed the copper, *not* on the Crown to show that it did). All agreed that a mine royal could be conveyed by the Crown by appropriate words, but that a general designation, as in the grant from Mary to the Earl, was not sufficient for this purpose.

In a note at the end of this case Plowden comments at length on the opinion of the three judges above cited, with which he seems to agree. From these comments I extract the following curious, though not specially relevant, passage:

"For if the value is not regarded, but the gold or silver, be it ever so little, shall entitle the Crown to the whole mine, from thence it would follow that the Crown would have all the mines of base metal in the realm, if that be true which those who are skilled *in re metallica* have written. For (as the ancient authors in this art affirm) there are but six kinds of metals in the earth, viz: gold, silver, tin, copper, lead, and iron, ... which, in Latin, they call *corpora metallica* and (as Eucelius says) '*constant ex sulphure patre, et argento vivo, matre.*' And afterwards he says that gold is to sulphur and quicksilver *quasi filius et perfectissimum metallum naturæ, et paululum immutata commixtione argenti vivi, fit argentum, quasi filia ignobilior fratre auro.*

"And the other metals, viz.: tin, copper, lead, and iron, as imperfect metals, proceed from their said father and mother who were by accidents become imperfect, *veluti leprosi filii ex leprosis parentibus.*"

The effect of this decision was to leave a sense of insecurity and discontent in the minds of land-owners, which, as Bainbridge says,* the statutes of William and Mary were, doubtless, intended to remove. At the present time, the Crown still claims, in England, the ownership of all gold- and silver-mines; but when these metals are found merely as constituents of base-metal ores, the royal ownership extends only to the right to purchase such ores at a statutory price.†

The statute of George III. merely increases the purchase-price under the Crown's pre-emptive right on lead-ore from £9 to £25 per ton.

Without going further into the somewhat complicated course of English legislation on this subject, we may conclude that it did not furnish the precise model of the New York statute, into the history of which we are now inquiring.

Probable Purpose of the New York Law.—As I have already observed, we may conclude, on circumstantial evidence, that this law was framed to include in a general system the satisfaction of

* The passages are as follows (*The Law of Mines and Minerals*, pp. 129, 130):

"This decision occurred in the time of Queen Elizabeth, when the prerogative of the Crown was, perhaps, at its greatest height, and the opinion of the nine judges does not appear to have gained the acquiescence of more recent lawyers. In 1640, the opinion of fifteen leading counsel, amongst whom are the names of Glanvil, Herbert, Grimston, and Maynard, was taken upon the subject. These gentlemen were all of opinion that, although the gold or silver contained in the base metal of a mine in the lands of a subject be of less value than the base metal, yet if the gold or silver countervail the charge of refining it, this is a mine royal, and as well the base metal as the gold or silver in it belong to the prerogative of the Crown.

"In the time of Queen Elizabeth, a society was established on the part of the Crown for the management of royal mines, most probably in consequence of the decision reported by Plowden. Several rules were framed for its guidance, particularly in 1670. The opinion of the fifteen counsel before mentioned seems to have been generally adopted. But considerable difference of opinion still prevailed in many instances with respect to the actual fact of the royal metals bearing the charges of refinement. The royal refiners and assayers became either less skilful or dishonest. At length the great case of Sir Carberry Price occurred. This case produced repeated trials at bar and at *nisi prius*, and occasioned considerable excitement in almost all parts of the kingdom. Sir C. Price succeeded at last in effectually precluding the claims of the Crown, but the spirit of mining adventure threatened to become extinct from the vexations and uncertain state of the law. The right of entry in search of royal mines was oppressive in the extreme, for the clause of compensation inserted in the royal patents was usually disregarded, and any mine which might have been discovered at great expense, seemed liable to be claimed as a royal mine. Valuable mines were concealed, and there was universal distrust. Such a state of things called loudly for a legislative remedy. This remedy was at last afforded"—in the statutes of William and Mary.

† 1 *W. & M.*, c. 30, § IV.; 5 *W. & M.*, c. 6; 55 *Geo. III.*, c. 134.

certain individual petitioners. In one case, at least, and not improbably in all the others, these petitioners were the owners of the land on which the mines mentioned had been discovered. Obviously, the provisions of the law which favor the land-owner were those of immediate significance, and the reservations in favor of the State were probably inserted on general principles, in the line of English precedents, and were not opposed by the petitioners or their friends, because they did not affect the cases represented in the petitions. In other words, these people were probably citizens, and discoverers of mines of copper, lead or iron (tin being out of the question), which they wished to own beyond dispute. It seems likely that the Ulster County petitions, at least, referred to lead-mines, of which a number were discovered at an early day in that region. Whether the Dutchess County petitions mentioned copper, lead or iron, could perhaps be determined by an examination of the history of mining enterprises in that county. It seems to me unlikely that iron-ore deposits were the subject of the petitions, because there were not then, any more than there are now, in the State of New York, deposits valuable for the manufacture of iron which contain any considerable admixture of ores of other metals. Such occurrences as the siliceous limonites of Leadville, Colorado, carrying silver- and lead-ores, find no parallels in New York.

Pursuing the same line of plausible surmise, we may infer that these petitioners, in consenting to the condition that the ore of their mines should contain at least two-thirds of its value of the base metals specified, must have felt that they were on the safe side. As to the possibly large number of mines which might afterwards be discovered, and the title to which, under this condition, might be placed in doubt, they did not trouble themselves. When occasion should arise, the State might give such relief as justice or policy should demand.

So far as I have been able to discover, no such occasion ever arose, and the State of New York was left with this antiquated code, inoperative and unnecessary in most of its features. It seems to me that enlightened statesmanship would now abolish the whole of it, and give all mineral deposits, out and out, to the owner of the land. Neither the State nor any individual in it would be injured by this step.

GEOLOGICAL HISTORY OF THE YELLOWSTONE NATIONAL PARK.

BY ARNOLD HAGUE, U. S. GEOLOGICAL SURVEY.

(Utah and Montana Meeting, July, 1887.)

IN the short time allotted to me* I can only hope to present to you a brief sketch of the main geological features of the country which you are about to visit. My remarks must, of necessity, be more or less incomplete, as my desire is not so much to elucidate any special problem connected with the many interesting geological questions to be found here, as rather to offer such a general view of the region as will enable you, during your five days' trip through the Park, to understand clearly something of its physical geography and geology.

The Yellowstone Park is situated in the extreme northwestern portion of the Territory of Wyoming. Its boundaries, as determined by the original Act of Congress setting apart the Park, are very ill-defined. At the time of the enactment of the law establishing this national reservation, the region had been but little explored, and its relation to the physical features of the adjacent country was but little understood. Since that time, surveys have shown that only a narrow strip, about two miles in width, was situated in the Territory of Montana, but it was also found that a still narrower strip extended westward into the Territory of Idaho. The question of properly establishing the boundaries, based upon our present knowledge of the country, is now before Congress, and an Act has already passed the Senate, proposing to make the northern boundary coincide with the boundary between Wyoming and Montana, and the western boundary coincide with the Wyoming and Idaho line. The Act under consideration extends the southern boundary of the Park to the 44th parallel of latitude, carrying the area of the reservation southward nine and one-half miles. The eastern boundary is made to coincide with the meridian of 109° 30′, adding a strip of country about twenty-four and one-half miles in width along the entire eastern side of the Park.

* This paper was presented as an address, given at a special session of the Institute, at Mammoth Hot Springs, Wyoming, on the borders of the National Park.

The area of the Park, as at present defined, is somewhat more than 3000 square miles, and the proposed addition increases the reservation by nearly 2000 square miles. The Park plateau, with the adjacent mountains, presents a sharply defined region, in strong contrast with the rest of the northern Rocky Mountains. It stands out boldly by itself, unique in topographical structure, and complete as a geological problem.

The central portion of the Yellowstone Park is, essentially, a broad, elevated, volcanic plateau, between 7000 and 8500 feet above sea-level, and with an average elevation of about 8000 feet. Surrounding it on the south, east, north, and northwest, are mountain ranges with culminating peaks and ridges rising from 2000 to 4000 feet above the general level of the enclosed table-land.

For present purposes it is needless to confine ourselves strictly to legal boundaries, but rather to consider the entire region in its broader physical features. It is worthy of note, however, that by the proposed enlargement the protected area will agree closely with the geographical province.

South of the Park, the Tetons stand out prominently above the surrounding country, the highest, grandest peaks in the northern Rocky Mountains. The eastern face of this mountain mass rises with unrivalled boldness for nearly 7000 feet above Jackson Lake. Northward, the ridges fall away abruptly beneath the lavas of the Park, only the outlying spurs coming within the limits of the reservation. For the most part the mountains are made up of coarse crystalline gneisses and schists, probably of Archæan age, flanked on the northern spurs by Paleozoic sediments.

To the eastward, across the broad valley of the Upper Snake, generally known as Jackson Basin, lies the well-known Wind River Range, famous from the earliest days of the Rocky Mountain trappers. The Northern end of this range is largely composed of Mesozoic strata, single ridges of Cretaceous sandstone penetrating still farther northward into the regions of the Park, and protruding above the great flows of lava.

Along the entire eastern side of the Park stretches the Absaroka Range—so-called from the Indian name of the Crow Nation. The Absaroka Range is intimately connected with the Wind River, the two being so closely related that any line of separation must be drawn more or less arbitrarily, based more upon geological structures and forms of erosion than upon physical limitations.

The Absarokas offer, for more than eighty miles, a bold, unbroken barrier to all western progress; a rough, rugged country, dominated by high peaks and crags from 10,000 to 11,000 feet in height. Only a few adventurous hunters and mountaineers cross the range by one or two dangerous, precipitous trails known to but few. The early trappers found it a forbidding land; prospectors who followed them, a barren one.

At the northeast corner of the Park a confused mass of mountains connects the Absarokas with the Snowy Range. This Snowy Range shuts in the Park on the north, and is an equally rough region of country, with elevated mountain masses covered with snow the greater part of the year, as the name would indicate. Only the southern slopes, which rim in the Park region, come within the limit of our investigation. Here the rocks are mainly granites, gneisses, and schists, the sedimentary beds, for the most part, referable to the pre-Cambrian series.

The Gallatin Range encloses the Park on the north and northwest. It lies directly west of the Snowy, only separated by the broad valley of the Yellowstone River. It is a range of great beauty, of diversified forms, and varied geological problems. Electric Peak, in the extreme northwestern corner of the Park, is the culminating point in the Range, and affords one of the most extended views to be found in this part of the country. Archæan gneisses form a prominent mass in the range, over which occur a series of sandstones, limestones, and shales, of Paleozoic and Mesozoic age, representing Cambrian, Silurian, Devonian, Carboniferous, Trias, Jura, and Cretaceous.[1] Immediately associated with these sedimentary beds, are large masses of intrusive rocks, which have played an important part in bringing about the present structural features of the range. They are all of the andesitic type, but showing considerable range in mineral composition, including pyroxene, hornblende, and hornblende-mica varieties. These intrusive masses are found in narrow dikes, in immense interbedded sheets forced between the different strata, and as laccolites, a mode of occurrence first described from the Henry Mountains in Utah, by Mr. G. K. Gilbert, but now well recognized elsewhere in the northern Cordillera.

We see, then, that the Absarokas rise as a formidable barrier on the eastern side of the Park, the Gallatins as a steep mural face on the west side, while the other ranges terminate abruptly, rimming in the Park on the north and south, and leaving a depressed region not unlike the parks of Colorado, only covering a more extended

area with a relatively deeper basin. The region has been one of profound dynamic action, and the centre of mountain building on a grand scale. On the accompanying map of the Yellowstone Park, which shows the position of the principal objects of interest, the relations of the ranges to the plateau are clearly indicated.

It is not my purpose at the present time to enter upon the details of geological structure of these ranges, each offering its own special study and field of investigation. My desire is simply to call your attention to their general features and mutual relations. So far as their age is concerned, evidence goes to show that the action of upheaval was contemporaneous in all of them, and coincident with the powerful dynamic movements which uplifted the north and south ranges, stretching across Colorado, Wyoming, and Montana. This dynamic movement blocked out, for the most part, the Rocky Mountains, near the close of the Cretaceous, although there is good reason to believe that in this region profound faulting and displacement continued the work of mountain building well into the Middle Tertiary period.

Throughout Tertiary time in the Park area, geological history was characterized by great volcanic activity, enormous volumes of erupted material being poured out in the Eocene and Middle Tertiary continuing with less force through the Pliocene, and extending into Quaternary time. Within very recent times there is no evidence of any considerable outburst; indeed, the region may be considered long since extinct. These volcanic rocks present a wide range in chemical and mineral composition and physical structure. They may all, however, be classed under three great groups—andesites, rhyolites, and basalts—following each other in the order named. In some instances, eruptions of basalt occurred before the complete extinction of rhyolite, but in general, the relative age of each group is clearly and sharply defined, the distribution and mode of occurrence of each presenting characteristic and salient features frequently marked by periods of erosion.

Andesites are the only volcanic rocks which have played an important part in producing the present structural features of the mountains surrounding the Park. As already mentioned, they occur in large masses in the Gallatin Range, while most of the culminating peaks in the Absarokas are composed of compact andesites and andesitic breccias. On the other hand, the andesites are not confined to the mountains, but played an active rôle in filling up the interior basin. That the duration of the andesitic eruptions was long

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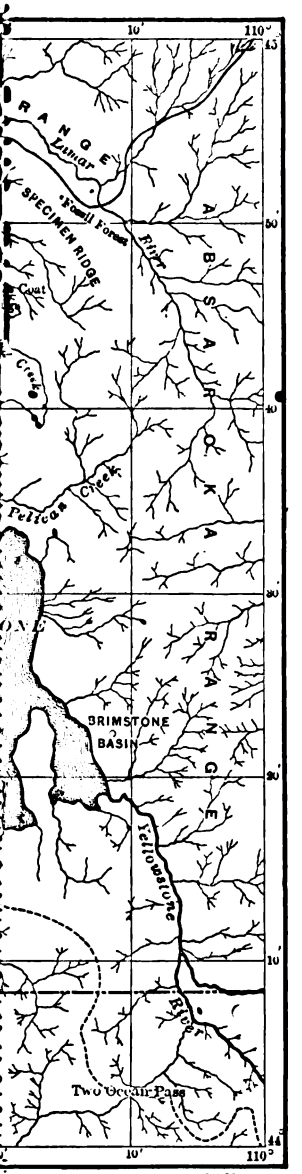

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er beds.

In early Tertiary times, a volcano burst forth in the northeast corner of the depressed area encircled by the Park Mountains, not far from the junction of the Absaroka and Snowy Ranges. While not to be compared in size and grandeur with the volcanoes of California and the Cascade Range, it is, for the Rocky Mountains, one of no mean proportions. It rises from a base about 6500 feet above sea-level, the culminating peak attaining an elevation of 10,000 feet. This gives a height to the volcano of 3500 feet from base to summit, measuring from the Archæan rocks of the Yellowstone Valley to the top of Mount Washburne. The average height of the crater rim is about 9000 feet above sea-level, the volcano measuring 15 miles across the base. The eruptive origin of Mount Washburne has long been recognized, and it is frequently referred to as a volcano. It is, however, simply the highest peak among several others, and represents a later outburst which destroyed in a measure the original rim and form of the older crater. The eruptions for the most part were basic andesites. Erosion has so worn away the earlier rocks, and enormous masses of more recent lavas have so obscured the original form of the lava-flows, that it is not easy for the inexperienced eye to recognize a volcano and the surrounding peaks as the more elevated points in a grand crater wall. By following around on the ancient andesitic rim, and studying the outline of the old crater, together with the composition of its lavas, its true origin and history may readily be made out. This older crater has, as yet, received no special designation, but when our maps and reports are finally published, this ancient geological ruin will receive an appropriate designation. This old volcano of early Tertiary time occupies a prominent place in the geological development of the Park, and dates back to the earliest outbursts of lava which have in this region changed a depressed basin into an elevated plateau. We have here a volcano situated far inland, in an elevated region, in the heart of the Rocky Mountains. It lies on the eastern side of the continent, only a few miles from the great continental divide which sends its waters to both the Atlantic and Pacific.

After the dying out of the andesitic lavas, followed by a period of erosion, immense volumes of rhyolite were erupted, which not only threatened to fill up the crater but to bury the outer walls of the volcano. On all sides the andesitic slopes were submerged beneath the rhyolite to a height of from 8000 to 8500 feet. This enormous mass of rhyolite, poured out after the close of the andesitic period, did more than anything else to bring about the present physical feat-

ures of the Park table-land. A tourist making the customary trip through the Park, visiting all the prominent geyser basins, hot springs, and the Grand Cañon and Falls of the Yellowstone, is not likely to come upon any other rock than rhyolite, excepting, of course, deposits from the hot springs. If he extended his journey to the lake-region, taking in Shoshone, Lewis, and Yellowstone lakes, and spending a week or ten days going over the beaten routes of travel, he will not, unless he ascends Mount Washburne, leave the rhyolite lavas. A description of the rhyolite region is essentially one of the Park plateau. Taking the bottom of the basin at 6500 feet above sea-level, these acidic lavas were piled up until the accumulated mass measured 2000 feet in thickness. It completely encircled the Gallatin Range, burying its lower slopes on both the east and west sides; it banked up all along the west flanks of the Absarokas, and buried the outlying spurs of the Teton and Wind River Ranges.

The Park Plateau covers an area approximately 50 by 40 miles, with a mean altitude of 8000 feet. It is accidented by undulating basins of varied outline and scored by deep cañons and gorges. Strictly speaking, it is not a plateau, at least it is by no means a level area, but a rugged country, characterized by bold escarpments and abrupt edges of mesa-like ridges. But few large vents or centers of volcanic activity for the rhyolite, have been recognized, the two principal sources being the volcano to which reference has already been made, and Mount Sheridan in the southern end of the Park. Mount Sheridan is the most commanding peak on the plateau, with an elevation of 10,200 feet above sea-level and 2600 feet above Heart Lake. From the summit of the peak on a clear day one may overlook the entire plateau country and the mountains which shut it in, while almost at the base of the peak lie the magnificent lakes which add so much to the quiet beauty of the region, in contrast with the rugged scenery of the mountains. From no point is the magnitude and grandeur of the volcanic region so impressive. The lava-flows, bounded on the east by the Absarokas, extend westward not only across the Park but across the Madison Plateau and out on to the great plains of Snake River, stretching far to the westward almost without a break in the continuity of the eruptive flows. Over the central portion of the Park, where the rhyolites are thickest, erosion has failed to penetrate to the underlying rock. Even such deep gorges as the Yellowstone, Gibbon and Madison Cañons have nowhere worn through these rhyolite flows. In the

Grand Cañon of the Yellowstone the andesitic breccias are found beneath the rhyolites, but the deepest cuts fail to reveal the underlying sedimentary beds. Although the rocks of the plateau for the most part belong to one group of acidic lavas, they by no means present the great uniformity and monotony in field appearance that might be expected. These 2000 square miles offer as grand a field for the study of structural forms, development of crystallization and mode of occurrence of acidic lavas as could well be found anywhere in the world. They vary from a nearly holocrystalline rock to one of pure volcanic glass. Obsidian, pumice, pitchstone, ash, breccia and an endless development of transition forms alternate with the more compact lithoidal lavas which make up the great mass of the rhyolite, and which in color, texture and structural development present an equally varied aspect. In mineral composition these rocks are simple enough. The essential minerals are orthoclase and quartz, with more or less plagioclase. Sanidine is the prevailing feldspar, although in many cases plagioclase forms occur nearly as abundantly as orthoclase. Chemical analyses, whether we consider the rocks from the crater of Mount Sheridan, the summit of the plateau, or the volcanic glass of the world-renowned Obsidian Cliff, present comparatively slight differences in ultimate composition.

The following analyses of two rocks, representing extreme forms in physical habit, show how closely they approach each other in composition of the original magma.

	No. 1. Madison Plateau.	No. 2. Obsidian Cliff.
Silica,	75.19	75.52
Titanic acid,	none	none
Phosphoric acid,	none	none
Alumina,	13.77	14.11
Ferric oxide,	0.61	1.74
Ferrous oxide,	1.37	0.08
Ferric sulphide,	.....	0.11
Manganese oxide,	trace	none
Lime,	0.68	0.78
Magnesia,	0.09	0.10
Lithia,	0.02	.....
Potash,	3.33	3.62
Soda,	3.83	3.93
Sulphuric acid,	0.29	.....
Water,	0.65	.....
Ignition,	.....	0.39
Total,	99.83	100.38

The rock from Madison Plateau was collected on the north side of Madison Cañon and was selected as a typical rock covering large areas of the Park. It is purplish-gray in color, rough in texture, porphyritic in structure, and characterized by well-developed sanidin and quartz. The obsidian, from Obsidian Cliff, is an excellent example of pure volcanic glass, wholly devoid of porphyritic crystals. In general the investigations of the laboratory confirm the observations of the field-geologist, that the differences exhibited by the volcanic product is not one of chemical or mineral composition, but rather of physical conditions under which the magma has cooled.

I have dwelt somewhat in detail upon the nature of these rocks for two reasons: *first*, because of the difficulty met with by the scientific traveller in recognizing the uniformity and simplicity of chemical composition of the rhyolite magma over the entire plateau, owing to its great diversity in superficial habit; *second*, on account of their geological importance in connection with the unrivalled display of the geysers and hot springs. That the energy of the steam and thermal waters dates well back into the period of volcanic action, there is, in my opinion, very little reason to doubt. As the energy of this underground heat is to-day one of the most impressive features of the country, I will defer commenting upon the geysers and hot springs until speaking of the present condition of the Park.

Although the rhyolite eruptions were probably of long duration and died out slowly, there is, I think, evidence to show that they occupied a clearly and sharply defined period between the andesites and basalt eruptions. Since the outpouring of this enormous body of rhyolite and building up of the plateau the region has undergone profound faulting and displacement, lifting up bodily immense blocks of lava and modifying the surface features of the country. Following the rhyolite came the period of basalt eruptions, which in comparison with the andesite and rhyolite eras was, so far as the Park was concerned, insignificant, both as regards the area covered by the basalt and its influence in modifying the physical aspect of the region. The basalt occurs as thin sheets overlying the rhyolite and in some instances as dikes cutting the more acidic rocks. It has broken out near the outer edge of the rhyolite body and occurs most frequently along the Yellowstone Valley, along the western foothills of the Gallatin Range and Madison plateau and again to the southward of the Falls River basin.

After the greater part of the basalt had been poured out came the glacial ice, which widened and deepened the pre-existing drainage-channels, cut profound gorges through the rhyolite lavas and modelled the two volcanoes into their present form. Over the greater part of the Cordillera of the central and northern Rocky Mountains, wherever the peaks attain a sufficiently high altitude to attract the moisture-laden clouds, evidences of the former existence of local glaciers are to be found. In the Teton Range several well-defined characteristic glaciers still exist upon the abrupt slopes of Mount Hayden and Mount Moran. They are the remnants of a much larger system of glaciers. The Park region presents so broad a mass of elevated country that the entire plateau was, in glacial times, covered with a heavy capping of ice. Evidences of glacial action are everywhere to be seen.

Over the Absaroka Range glaciers were forced down into the Lamar and Yellowstone Valleys, thence westward over the top of Mount Evarts to the Mammoth Hot Springs basin. On the opposite side of the Park the ice from the summit of the Gallatin Range moved eastward across Swan Valley and passing over the top of Terrace Mountain joined the ice-field coming from the east. The united ice-sheet plowed its way northward down the valley of the Gardiner to the Lower Yellowstone where the broad valley may be seen strewn with the material transported from both the east and west rims of the Park.

Since the dying out of the rhyolite eruptions erosion has greatly modified the entire surface features of the Park. Some idea of the extent of this action may be realized when we recall that the deep cañons of the Yellowstone, Gibbon and Madison Rivers—cañons in the strictest use of the word—have all been carved out since that time. To-day these gorges measure several miles in length and from 1000 to 1500 feet in depth.

To the geologist one of the most impressive objects on the Park Plateau is a transported boulder of granite which rests directly upon the rhyolite near the brink of the Grand Cañon about three miles below the Falls of the Yellowstone. It stands alone in the forest, miles from the nearest glacial boulder. Glacial detritus carrying granitic material may be traced upon both sides of the cañon wall, but not a fragment of an older rock, more than a few inches in diameter, has been recognized within a radius of many miles. This massive block, although irregular in shape and somewhat pointed towards the top measures 24 feet in length by 20 feet in breadth and stands

18 feet above the base. The nearest point from which it could have been transported is distant 30 or 40 miles. Coming upon it, in the solitude of the forest with all its strange surroundings, it tells a most impressive story. In no place are the evidences of frost and fire brought so forcibly together as in the Yellowstone National Park.

Since the close of the ice period no geological events of any moment have brought about any changes in the physical history of the region other than those produced by the direct action of steam and thermal waters. A few insignificant eruptions have probably occurred, but they failed to modify the broad outlines of topographical structure and present but little of general interest beyond the evidence of the continuance of volcanic action into quaternary times. Volcanic activity in the Park may be considered as long since extinct. At all events indications of fresh lava-flows within historical times are wholly wanting. This is not without interest, as evidence of underground heat may be observed everywhere throughout the Park in the waters of the geysers and hot springs. All our observations point in one direction and lead to the theory that the cause of the high temperatures of these waters must be found in the heated rocks below, and that the origin of the heat is in some way associated with the source of volcanic energy. It by no means follows that the waters themselves are derived from any deep-seated source; on the contrary, investigation tends to show that the waters brought up by the geysers and hot springs are mainly surface waters which have percolated downwards a sufficient distance to become heated by large volumes of steam ascending through fissures and vents from much greater depths. If this theory is the correct one, it is but fair to demand that evidence of long-continued action of hot waters and superheated steam should be apparent upon the rocks through which they passed on their way to the surface. This is precisely what one sees in innumerable places on the Park plateau. Indeed, the decomposition of the lavas of the rhyolite plateau have proceeded on a most gigantic scale, and could only have taken place after the lapse of an enormous period of time, and the giving off of vast quantities of heat, if we are to judge at all by what we see going on around us to-day. The ascending currents of steam and hot water have been powerful geological agents, and have left an indelible impression upon the surface of the country. The most striking example of this action is found in the Grand Cañon of the Yellowstone. From the lower falls for three miles down the river the abrupt

walls upon both sides of the cañon, a thousand feet in depth, present a brilliancy and mingling of color beyond the power of description. From the brink of the cañon to the water's edge, the walls are sheer bodies of decomposed rhyolite. Varied hues of orange, red, purple, and sulphur-yellow, are irregularly blended in one confused mass. There is scarcely a piece of unaltered rock in place. Much of it is changed into kaolin, but from rhyolite, still easily recognized, occur transition products of every possible kind to good porcelain clay. This is the result of the long-continued action of steam and vapors upon the rhyolite lavas. Through this mass of decomposed rhyolite the course of ancient steam vents in their upward passage may still be traced, while at the bottom of the cañon hot springs, fumaroles, and steam vents, are still more or less active, but probably with diminished power.

It is needless to weary you with the details of this decomposition, but I may add that investigations in the laboratory upon these transition-products fully substantiate field observations.

Still other areas are quite as convincing, if not on so grand a scale, as the Yellowstone Cañon. Joseph's Coat Basin, on the east side of the cañon, and Brimstone Hills, on the east side of the Yellowstone Lake, an extensive area on the slopes of the Absaroka Range, both present evidences of the same chemical processes brought about in the same manner. It is not stating it too strongly to say that the plateau on the east side of the Grand Cañon, from Broad Creek to Pelican Creek, is completely undermined by the action of superheated steam and alkaline waters on the rhyolite lava. The same processes may be seen going on to-day in all the geyser basins. To accomplish these changes a long period of time must have been required. The study of comparatively fresh vents show almost no change from year to year, although careful scrutiny during a period of five years detects a certain amount of disintegration, but infinitely small in comparison with the great bodies of altered rock. This is well shown in a locality like the Monarch Geyser in the Norris Geyser Basin, where the water is thrown out at regular intervals through a narrow fissure in the rock.

The Grand Cañon of the Yellowstone offers one of the most impressive examples of erosion, on a grand scale, within recent geological times. It is self-evident that the deep cañon must be of much later origin than the rock through which it has been worn, and it seems quite clear that the course and outlines of the cañon were in great part determined by the easily eroded decomposed material

forming the cañon walls, and this, in turn, was brought about by the slow processes just described.

The evidence of the antiquity of the hot spring deposits is, perhaps, shown in an equally striking manner, and by a wholly different process of geological reasoning. Terrace Mountain is an outlying ridge of the rhyolite plateau just west of the Mammoth Hot Springs. It is covered on the summit with thick beds of travertine, among the oldest portions of the Mammoth Hot Springs deposits. It is the mode of occurrence of these calcareous deposits from the hot waters which have given the name to the mountain. Lying upon the surface of this travertine on the top of the mountain are found glacial boulders brought from the summit of the Gallatin Range, fifteen miles away, and transported on the ice-sheet across Swan Valley and deposited on the top of the mountain, 700 feet above the intervening valley. It offers the strongest possible evidence that the travertine is older than the glacier which has strewn the country with transported material. How much travertine was eroded by the ice is, of course, impossible to say, but so friable a material would yield readily to glacial movement.

Still another method of arriving at the great antiquity of the thermal energy, and the age of the hot-spring formation, is by determining the rate of deposition and measuring the thickness of the accumulated sinter. This method, although the one which would perhaps first suggest itself, is, in my opinion, by no means as satisfactory as the geological reasoning already given. It is unsatisfactory because no uniform rate of deposition can be ascertained for even a single area, like the Upper Geyser Basin, and it is still more difficult to arrive at any conclusion as to the growth of the sinter in the past. Moreover, it is quite possible that heavy deposits may have suffered erosion before the present sinter was laid down. It, however, corroborates other methods, and possesses the advantage of being a direct way.

It may be well to add here that there exists the greatest contrast between the deposits of the Mammoth Hot Springs and those found upon the plateau. At the Mammoth Springs they are nearly pure travertine, with only a trace of silica, analyses showing from 95 to 99 per cent. of calcium carbonate. On the plateau, the deposits consist for the most part of siliceous sinter, locally termed "geyserite." The reason for the difference is this: At the Mammoth Hot Springs the steam, although ascending from fissures in the igneous rocks, comes in contact with the waters found in the Meso-

zoic strata, which here form the surface-rocks. The Jura or Cretaceous limestones have furnished the lime held in solution and precipitated on the surface as travertine. On the other hand, the mineral constituents of the plateau waters are derived almost exclusively from the highly acidic lavas, which, as it will be seen by reference to the analyses, carry but a small amount of lime.

Deposition of sinter from the hot waters of the geyser basins depends in a great measure on the amount of silica held in solution, which varies considerably at the different localities, and may have varied still more in past time. The silica, as determined by analyses, ranges from .22 to .60 grammes per kilogramme of water, the former being the amount found in the water of the caldron of the Excelsior Geyser, and the latter at the Coral Spring in the Norris Basin. Analysis shows that from one-fifth to one-third of the mineral matter held in solution consists of silica, the remaining constituents being readily soluble salts carried off by the surface drainage. A few springs highly charged with silica, like the Coral, deposit it on the cooling of the waters; but such springs, however, are exceptional, and I do not recall a single instance of a spring in the Upper Geyser Basin precipitating silica in this way. At most springs and geysers it results only after evaporation, and not from mere cooling of the water. It seems probable that the nature and amount of alkaline chlorides and carbonates present influence the separation of silica. Temperature also may in some degree influence the deposition. My friend, Mr. Elwood Hofer, one of the best guides to this region, and a keen observer of nature, has called my attention to an observation of his made in mid-winter, while on one of his snow-shoe trips through the Park. He noticed that certain overflow pools of spring-water, upon being frozen, deposited a considerable amount of mineral matter. He has sent me specimens of this material which, upon examination, proved to be identical with the silica deposited from the Coral Spring upon the cooling of the water. Demijohns of geyser water which have been standing for one or two years have failed to precipitate any silica. Quite recently, in experimenting upon these waters in the laboratory, it was noticed that on reducing them nearly to the freezing point no change took place, but upon freezing the waters there was an abundant separation of free silica. The waters frozen in this way were collected from the Coral Spring, Norris Basin, and the Taurus Geyser, Shoshone Basin.

Again, there is no doubt that the algeous growths found flour-

ishing in the hot waters of the Park favor the secretion of silica, and exert an influence in building up the geyserite far greater than one would at first be led to suppose. These low forms of vegetable life occur in nearly all pools, springs, and running-waters up to a temperature of 185° Fahr., only 13° below the boiling point, at the Upper Geyser Basin. If time permitted, much might be said on this subject. I will only add, that Mr. Walter H. Weed, in connection with his other duties on the Geological Survey, has devoted much time to a study of these algeous growths, and the results of his investigations will form an important chapter in the final publications.

Several methods have been devised for ascertaining the growth of deposition of the geyserite. One way is by allowing the water to trickle over twigs, dried grasses, or almost anything exposing considerable surface, and noting the amount of incrustration. This way gives the most rapid results, but is far from satisfactory, and by no means reproduces the conditions existing in nature. Other methods employed are, placing objects on the surface of the water, or, still better, partially submerging them in the hot pools, or again by allowing the water to run down an inclined plane with frequent intervals for evaporation and concentration.

The vandals who delight to inscribe their names in public places have invaded the geyser basins in large numbers, and left their addresses upon the geyserite in various places. It is interesting to note how quickly these inscriptions become indelible by the deposition of the merest film of silica upon the lead pencil marks, and, at the same time, how slowly they build up. Names and dates known to be six and eight years old remain perfectly legible, and still retain the color and lustre of the graphite. That there is some increase in the thickness of the incrustation is evident, although it grows with incredible slowness. Mr. Weed tells me that he has been able, in at least one instance, to chip off this siliceous film and reproduce the writing with all its original distinctness, showing conclusively that a slow deposition has taken place. Pencil inscriptions upon the siliceous sinter at Rotomahana Lake, in New Zealand, are said to be legible after the lapse of twenty or thirty years. It is easy to see that various ingenious devices might be planned to estimate the rate of deposition, but in my opinion none of them equal a close study of the conditions found in nature, especially where investigations of this kind can be watched from year to year. All observations show an exceedingly slow building up of the geyserite formation. This

is well seen in the repair going on where the rims surrounding the hot pools have been broken down, and where it might be supposed that the building-up process was under the most favorable conditions, yet, in a number of instances, I can see no appreciable change in three or four years. Revisiting hot springs in out-of-the-way places after several years absence, I am surprised to see that objects that I had noted carefully at the time remained unchanged. Taking the entire area of the Upper Geyser Basin covered by sinter, I believe that the development of the deposit does not exceed one-thirtieth of an inch a year, and this estimate I believe to be much nearer the maximum than the minimum rate of growth. The thickness of the geyserite has never been ascertained; the greatest thickness measured is 70 feet, the depth reached in the conduit of "Old Faithful" geyser, without meeting any obstruction. Supposing the deposit to have been built up with the same slowness as that going on to-day, and assuming it to grow at the rate given—one-thirtieth of an inch a year—it would require over 25,000 years to reach its present development. This gives us a great antiquity for the geyserite, but I believe that the deposition of the siliceous sinter in the Park has been going on for a still longer period of time. It is certain that the decomposition of the rhyolite of the plateau dates still further back.

From a geological point of view, there is abundant evidence that thermal energy is gradually becoming extinct. Tourists revisiting the Park, after an absence of two or three years, occasionally allude to the springs and geysers as being less active than formerly, and as showing indications of rapidly dying out. It is true that slight changes are constantly taking place, that certain springs become extinct or discharge less water, but this action is fully counterbalanced by increased activity in other localities. Close examination of the source of the thermal waters fails to detect any diminution in the supply. Moreover, it stands to reason that if the flow of these waters dated, geologically speaking, far back into the past, that the few years embraced within the historical records of the Park would be unable to indicate any perceptible change based upon a gradual diminution of the heat. Accurate descriptions of the region go back only to 1871, the year of the first exploration by Dr. F. V. Hayden.

The number of geysers, hot springs, mud-pots and paint-pots scattered over the Park exceeds 3500, and if to these be added the fumaroles and solfataras from which issue in the aggregate enormous volumes of steam and acid and sulphur vapors, the number of active

vents would in all probability be doubled. Each one of these vents is a center of decomposition of the acid lavas. In the four principal geyser basins the geysers in action or known to have been active within the past sixteen years number 84. The following list comprises all the geysers that are known in the Norris, Lower, Midway, and Upper Geyser Basins.

NORRIS GEYSER BASIN.—14.

Arsenic,	Growler,	Pebble,
Constant,	Hurricane,	Schlammkessel,
Echinus,	Minute,	Veteran,
Fearless,	Monarch,	Vixen.
Fissure,	Pearl,	

LOWER GEYSER BASIN.—17.

Bead,	Impulsive,	Spasm,
Clepsydra,	Jet,	Spray,
Conch,	Mound,	Steady,
Fountain,	Narcissus,	Surprise,
Fitful,	Pink Cone,	White Dome.
Great Fountain,	Rosette,	

MIDWAY GEYSER BASIN.—4.

Excelsior,	Flood,	Rabbit,	Tromp.

UPPER GEYSER BASIN.—40.

Artemesia,	Giantess,	Saw Mill,
Bee Hive,	Grand,	Sentinel,
Bijou,	Grotto,	Shell,
Bonita,	Infant,	Spasmodic,
Brilliant,	Jewel,	Spanker,
Bulger,	Liberty,	Spiteful,
Cascade,	Lion,	Splendid,
Castle,	Lioness,	Sponge,
Catfish,	Mastiff,	Spouter,
Churn,	Midget,	Sprinkler,
Cliff,	Model,	Sprite,
Comet,	Mortar,	Tardy,
Cubs,	Oblong,	Three Sisters,
Daisy,	Old Faithful,	Triplets,
Fan,	Restless,	Turban,
Giant,	Riverside,	White.
	Rocket,	

A comparative study of the analyses of the fresh rhyolite, the various transition-products and the thermal waters, points clearly to the fact that the solid contents of these waters are derived for the

most part from the volcanic rocks of the plateau. During the progress of the work of the Geological Survey in the Yellowstone Park, there have been collected from nearly all the more important localities samples of the waters, which have been subjected to searching chemical analyses in the laboratory of the Survey by Messrs. F. A. Gooch and J. E. Whitfield, the results of whose work will be published at an early date.

The following analyses of hot waters from the three principal geyser basins, serve to shown their chemical composition:

	Constant Geyser.		Hygeia Spring.		Old Faithful.	
	Grams. per kilo. of water.	Per cent. of total mat. in solution.	Grams. per kilo. of water.	Per cent. of total mat. in solution.	Grams per. kilo. of water.	Per cent. of total mat. in solution.
Silica	0.4685	28.88	0.2477	20.98	0.3828	27.52
Sulph. acid	0.0923	5.69	0.0196	1.65	0.0152	1.09
Carbonic acid	0.0155	0.95	0.2907	24.62	0.0894	6.43
Boracic acid	0.0317	1.95	0.0239	2.02	0.0148	1.07
Arsenious acid	0.0018	0.11	0.0034	0.29	0.0021	0.15
Chlorine	0.5740	35.39	0.2487	21.06	0.4391	31.57
Bromine	trace	.....	trace	.....	0.0034	0.25
Hydr. sulph.	none	.....	none	.....	0.0002	0.01
Oxygen (basic)	0.0185	1.14	0.0504	4.27	0.0419	3.02
Iron	trace	.....	none	.....	trace	.....
Manganese	none	.....	none	.....	trace	.....
Aluminium	0.0048	0.29	0.0036	0.31	0.0009	0.06
Calcium	0.0146	0.90	0.0064	0.54	0.0015	0.11
Magnesium	0.0018	0.11	0.0022	0.19	0.0006	0.04
Potassium	0.0745	4.60	0.0154	1.30	0.0267	1.92
Sodium	0.3190	19.67	0.2654	22.48	0.3666	26.36
Lithium	0.0030	0.19	0.0032	0.27	0.0056	0.40
Ammonium	0.00127	0.08	0.00021	0.02	0.00001	.....
Hydr. (HCl)	0.0008	0.05	.....	.....	.....	.....
Caesium	.....	.....	.....	.....	trace	.....
Rubidium	.....	.....	.....	.....	trace	.....
Totals	1.62207	100.00	1.18081	100.00	1.39081	100.00

Constant Geyser, Norris Geyser Basin—Date of collection, September 13, 1885; temperature, 198° F.; reaction, slightly acid; specific gravity. 1.0011.

Hygeia Spring, Lower Geyser Basin—Date of collection, September 11, 1885; temperature, 109° F.; reaction, alkaline; specific gravity, 1.0010.

Old Faithful Geyser, Upper Geyser Basin—Date of collection, September 1, 1884; temperature, 190° F.; reaction, alkaline; specific gravity, 1.00096.

They are all siliceous alkaline waters holding the same mineral constituents but in varying quantities. Silica forms the principal deposit not only immediately around the springs, but over the entire

floor of the basins. The carbonates, sulphates, chlorides and traces of other easily soluble salts are carried off in the waters. Oxides of iron and manganese and occasionally some calcite occur under certain conditions in the caldrons of the hot springs or immediately around their vents. Concentrations from large quantities of these waters fail to show the presence of even a trace of copper, silver, tin or other metal. Nearly all the waters carry arsenic, the amount present, according to Messrs. Gooch and Whitfield, varying from .02 to .25 per cent. of the mineral matter in solution.

Among the incrustations found at several of the hot springs and geysers is a leek-green amorphous mineral, which proves on investigation to be scorodite, a hydrous arseniate of iron. The best occurrence observed is at Joseph's Coat Springs, on the east side of the Grand Cañon of the Yellowstone, where it occurs as a coating upon the siliceous sinter lining the caldron of a boiling spring. Analysis shows a nearly pure scorodite, agreeing closely with the theoretical composition.

Ferric oxide, ...	34.94
Arsenic acid, ...	48.79
Water, ...	16.27
	100.00

Alteration of the scorodite into limonite takes place readily, which in turn undergoes disintegration by the wearing of the water, and is mechanically carried away. So far as I know this is the only occurrence where scorodite has been recognized as deposited from the waters of thermal springs. Although pure scorodite is only sparingly preserved at a few localities in the Yellowstone Park, it is easily recognized by its characteristic green color, in strong contrast with the white geyserite and yellow and red oxides of iron. After a little practice the mineral green of scorodite is not easily mistaken for the vegetable green of the algeous growths. The latter is associated everywhere with the hot waters, while the former, an exceedingly rare mineral, is obtained only in small quantities after diligent search. In America traces of arsenic have been reported from several springs in Virginia and quite recently sodium arseniate has been detected in the hot springs of Ashe County, North Carolina. Arsenical waters of sufficient strength to be beneficial for remedial purposes and not otherwise deleterious are of rare occurrence. In France the curative properties of arsenical waters have long been recognized, and the famous sanitarium of La Bour-

boule in the volcanic district of the Auvergne has achieved a wide reputation for the efficacy of its waters in certain forms of nervous diseases. Hygeia Spring, supplying the bath houses at the hotel in the Lower Geyser Basin, carries .3 of a grain of sodium arseniate to the gallon. The Yellowstone Park waters, while they carry somewhat less arsenic than those of La Bourboule, greatly excel the latter in their enormous overflow. It is stated that the entire discharge from the springs of La Bourboule, amounts to 1500 gallons per minute. The amount of hot water brought to the surface by the hot springs throughout the Park is by no means easily determined, although during the progress of our investigations we hope to make an approximate estimate. Some idea of the amount of hot water brought to the surface and carried off by the great drainage channels may be formed at the Midway Basin. According to the most accurate measurements which could be made, the discharge from the caldron of the Excelsior Geyser into the Firehole River during the past season amounted to 4400 gallons of boiling water per minute, and there is no evidence that this amount has varied within the last two or three years. The sample of the Excelsior Geyser water collected August 25th, 1884, yielded .19 grains of sodium arseniate to the gallon. It is impossible to say as yet what curative properties these Park waters may possess in alleviating the ills of mankind. Nothing but an extended experience under proper medical supervision can determine. I may say that no hot springs with which I am acquainted prove so delightful for bathing purposes and so agreeable in their action upon the skin.

Changes modifying the surface-features of the Park in recent times are mainly those brought about by the filling up with detrital material from the mountains, the valleys and depressions worn out by glacial ice, and those produced by the prevailing climatic conditions. Between the Park country and what is known as the arid regions of the west there is the greatest possible contrast. Across the Park Plateau and the Absaroka Range the country presents a continuous mountain mass seventy-five miles in width, with an average elevation unsurpassed by any area of equal extent in the northern Rocky Mountains. It is exceptionally situated to collect the moisture-laden clouds, which coming from the southwest precipitate immense quantities of snow and rain upon the cooled table-land and neighboring mountains. The climate in many respects is quite unlike that found in the adjacent country, as is shown by the meteorological records, the amount of snow and rainfall being higher, and the mean annual

temperature lower. Rain storms occur frequently throughout the summer, while snow is quite likely to fall any time between September and May. Protected by the forests the deep snows of winter lie upon the plateau well into mid-summer, while at still higher altitudes, in sheltered places, it remains throughout the year. By its topographical structure the Park is designed by nature as a reservoir for receiving, storing and distributing an exceptional water supply, unexcelled by any area near the headwaters of the great continental rivers. The Continental Divide, separating the waters of the Atlantic from those of the Pacific, crosses the Park plateau from southeast to northwest. On both sides of this divide lie several large bodies of water which form so marked a feature in the scenery of the plateau that the region has been designated the lake country of the Park. Yellowstone Lake, the largest lake in North America at this altitude (7740 feet) and one of the largest in the world at so high an elevation above sea-level, presents a superficial area of 139 square miles, and a shore-line of nearly 100 miles. From measurements made near the outlet of the lake in September, 1886, the dryest period of the year, the discharge was found to be 1525 cubic feet per second, or about 34,000,000 imperial gallons per hour.

At the same time all the principal lakes and streams in the Park were carefully gauged. Dr. William Hallock, who undertook this work, estimated that the amount of water running into the Park and leaving it by the Yellowstone, Gallatin, Madison, Snake and Falls Rivers, the five main drainage channels, would be equivalent to a stream 5 feet deep, 190 feet wide, with a current of 3 miles per hour, and that over an area of 4000 square miles the minimum discharge was equal to one cubic foot per second per square mile. For the preservation and regulation of this water supply the forest, which covers the mountains, valleys and table-lands, and everywhere borders upon the lake shores, is of inestimable value. Of the present Park area about 84 per cent. is forest-clad, almost wholly made up of coniferous trees. The timber is by no means of the finest quality, but for the purpose of water protection it meets every possible requirement. Much has been said of late years by scientific and experienced persons of the great necessity of preserving the forests near the sources of our great rivers. It is mainly for the forest protection that the proposed enlargement is demanded by the public welfare. In my opinion no region in the Rocky Mountains is so admirably adapted for a forest reservation as the Yellowstone National Park.

STRUCTURAL RELATIONS OF ORE-DEPOSITS.

BY S. F. EMMONS, U. S. GEOLOGICAL SURVEY, WASHINGTON, D. C.

(Boston Meeting, February, 1888.)

"The obscurity which still veils from us the true nature of veins will become more and more cleared up when they can be considered in connection with the geological structure of the regions in which they occur."*

THESE words, according to Lossen, the biographer of the late Dr. A. von Groddeck, of Clausthal, were the last received by him from that eminent man, perhaps the most far-sighted and carefully progressive of recent writers on ore-deposits. They seem to me to form a fitting text for what I am about to say; since I find that many of the conclusions at which I have arrived by my own study of the structural relations of ore-deposits, or, in other words, of the structural and dynamic geology of the regions in which they occur in this country, have been similarly determined by him within recent years in Europe, and especially in his home field of work, the Hartz.

I shall not enter upon a detailed account of how far other workers may have arrived at similar conclusions with myself, since these questions of priority are of little interest, except to those immediately concerned; but will briefly present the methods by which, in my work in this country, I have arrived at certain generalizations, and will point out the practical bearing which they may have upon the work of mining engineers.

In my first thorough and systematic examination of an important group of ore-deposits, viz., that of Leadville, Colorado, I was enabled, through the generous facilities afforded me by the then Director of the Geological Survey, Mr. Clarence King, to make a most elaborate and detailed study of the geology, not only of the immediate vicinity of the mines, but of the adjacent mountain regions, based upon accurate topographical and underground maps, and

* "Das Dunkel, welches uns die wahre Natur der Gänge noch immer verhüllt, wird sich mehr und mehr lichten wenn sie im Zusammenhang mit dem geognostischen Bau der Gegenden, in denen sie auftreten, betrachtet werden können."

thereby to determine the structural relations of these ore-bodies with an exceptional degree of detail and accuracy. The inductions based upon these structural relations have now borne the test of seven years of active exploration by a most energetic and enterprising mining community, and their substantial accuracy has been so abundantly proved as to afford an excellent practical illustration of the importance, in this class of deposits, of the structural study.

In entering upon this work I was, I must admit, far from familiar with all that had been written upon the theory of ore-deposits, and I purposely refrained from reading up upon the subject until my field-work was completed, in order to avoid the danger of any unconscious bias which might influence my interpretation of the facts of nature. It was commonly said at that time of the Leadville deposits "that they went contrary to all the theories of geologists." Such statements are in general, I find, liable to originate in a misapprehension, either of the true conditions which prevail at the locality in question, or of what are the best geological theories. In this case I found, in two respects at least, that the process of ore-deposition at Leadville was not in accordance with the most widely accepted theories upon the subject. First, they were not the filling of pre-existing open cavities or caves in the limestone, but had been formed by a gradual replacement of the rock-material by the substances brought in by the ore-bearing solutions. Secondly, these solutions did not come directly from below, as seemed to be considered a necessary condition for the deposition of ores, but in this case could be proved to have reached the immediate *locus* of deposit from above.

As I pointed out in a paper read before the Institute two years since (*Trans.*, xv. 125), in stating the results I had arrived at in regard to the genesis of the Leadville deposits, I made no claim of presenting any complete general theory or classification of ore-deposits at that time, nor have my observations been yet sufficiently wide to enable me to do so now. I did, however, conclude, as a result of my investigations, that existing theories and classifications were inadequate to account for the various forms under which ores are found in nature, and that this inadequacy arose probably from the want of some common underlying genetic basis. Far too few of the descriptions of ore-deposits in various parts of the world, upon which the existing theories have been founded, are based upon accurate determinations of their structural relations. I have, therefore, been emboldened to assume that, by giving more attention to

these structural relations than has hitherto been done, many existing uncertainties might be cleared up, and perhaps, in time, a rational or natural classification of ore-deposits might be formed. In the paper above mentioned I stated the conclusions I had been enabled to form from my Leadville and later studies in regard to the genesis of certain ore-deposits, mainly those in limestone, and touched lightly on their probable application to what are known as fissure-veins, of which I had, up to that time, had but limited opportunities of personal observation. Since then, however, especially during the past summer, I have visited many so-called fissure-deposits of great variety of type, and during these visits I have been so struck with the uniform occurrence of certain elementary structural conditions that I have ventured to make some preliminary generalizations based upon them, that seem likely to find a very wide practical application, and to furnish a basis, with modifications that may be afterwards introduced as a result of wider observation, for a general classification of ore-deposits. Whether this prove to be the case or not, it seems that this method of regarding ore-deposits, while I can hardly claim for it much that is absolutely new, has the merit of being a simple, rational method, which obviates many of the uncertainties and misapprehensions, especially in practical application, that constantly arise from the more complicated existing systems.

Preliminary statements.—The ore-deposits which I propose to consider are original or primary deposits that have been formed later than the enclosing rock. They exclude, therefore, secondary deposits, whether of mechanical or detrital origin, like placers, or of chemical origin, like bog-iron ores, which result from the superficial leaching of other deposits. They also exclude deposits, such as beds of coal, gypsum and the like, which have been deposited, so to speak, contemporaneously with the enclosing rock.

Even with these exclusions they include practically all the workable primary deposits of useful metals; for I have yet to see such deposits which can be proved to have been formed contemporaneously with the enclosing rocks. It is possible that in some cases, as Irving holds with regard to the Lake Superior iron-deposits, that there was an original nucleus of ore contemporaneously precipitated with the sediments which formed the rocks, but which has been added to to such an extent by percolating solutions replacing the adjoining portions of the enclosing rocks, that the present ore-body is practically a later-formed deposit.

I believe, with Pumpelly, Pošepnỳ, and others, that all our workable deposits of the useful metals are the result of a process or series of processes of concentration or aggregation of material previously disseminated in a minute form through the mass of the rocky crust of the earth.

With regard to these deposits there are certain elementary postulates, which at the risk of apparent unnecessary repetition from former papers I think it best to briefly enumerate.

1st. That these deposits were formed from aqueous solutions. In other words, they have been gathered up by waters containing varying substances in solution, and percolating through the earth's crust, and again deposited in their present concentrated form under a change of conditions which favored precipitation rather than solution.

It seems hardly necessary at the present day to bring forward any arguments in favor of the aqueous deposition of ores as opposed to the notion of an eruptive origin, which at the best has been a theoretical view not founded upon carefully and critically determined facts; but if there exist any lingering doubts in any mind as to the capability of solutions to take up and redeposit the various minerals found in our ore-deposits I would recommend a perusal of Daubrée's recent voluminous work upon subterranean waters (*Les eaux souterraines à l' époque actuelle et aux époques anciennes*).

2d. That, under given conditions of heat and pressure, all substances are more or less permeable to water. Hence we are justified in assuming that, at sufficient depths within the earth's crust, waters circulate or permeate all rocks, even those apparently impermeable, and thus may dissolve out minute quantities of the more readily attackable materials within their mass, and remove them to some other place. It is evident, however, that these waters will constantly tend to concentrate in such portions of the rocky mass as offer a more ready passage or flow, and that during such passage or flow they will seek relief from heat or pressure, and hence come into differing conditions which may favor precipitation, deposition, or chemical interchange of the materials they carry in solution.

Among the determining causes of solution and of precipitation or deposition, the chemical forces undoubtedly play an important part. The capacities of various alkaline solutions in taking up metallic minerals, especially when hot, and the conditions of diminished heat and pressure, or of dilution, which would favor precipitation of these minerals, present an extremely interesting and important subject of discussion; but it is one that would take too much

time to treat within the limits of this paper. I shall, therefore, confine myself as far as possible to the mechanical or structural conditions which would favor ore-deposition; in other words, consider what are the natural water-channels within the earth's crust, and what forms of ore-deposits such water-channels will give rise to, when the chemical and other conditions are such as to favor a deposition of valuable minerals in workable quantity by them.

3d. A further postulate, which I insist on the more strenuously for the reason that in former times the opposite view has been so almost universally held among writers on ore-deposits, is that a large pre-existing open cavity is not a necessary condition for ore-deposition. In other words, that ore-deposits are to a large extent the actual replacement of the country rock by vein-materials, and that the filling up by these materials is rather that of interstitial spaces than of what might properly be considered open cavities. I by no means deny that a certain limited amount of unoccupied space may have existed in the water-channel previous to the ore-deposition, but I wish to remove the tendency to misapprehension caused by the previously conceived view, under which, for instance, one might regard the whole width of an 8-foot vein as material brought in from a foreign source, when in point of fact more than seven feet of it might be more or less completely altered or replaced country rock, and less than a foot (and this not in one body) represent spaces actually filled in. I can even conceive of cases of veins showing what is supposed to be the comb-structure, from which the idea of the pre-existing open space was originally derived, where this structure is produced by sheeting of the country-rock, the different sheets being replaced by different mineral combinations, as will be explained later on.

Let us consider, then, what would be the natural and most readily available channels which would tend to collect the waters circulating within the earth's crust, and constitute their primary water-courses.

It is evident that the nearest analogy to the flow of underground waters which can actually come under our observation will be found in that of springs, especially that of thermal springs. A study of the phenomena connected with spring-action and deposition, such as has been made by Daubrée in his valuable work above cited, materially aids our conceptions of what may have taken place in the deeper-seated channels which have been in former geological periods the scene of ore-deposition.

One must beware of following this analogy blindly, however, and bear in mind that most ore-deposits must have been formed at considerable depths below the surface, and brought to the comparatively short distance from that surface which they now occupy by upheaval and erosion.

An instance of the errors into which one may be otherwise led is afforded by a paper read before the Institute two years since by Prof. T. B. Comstock* on the "Geology and Vein-Structure of Southwestern Colorado.

Owing to the vagueness of Mr. Comstock's statements, and his many misconceptions of geological facts, we cannot always be sure of his meaning; but from a careful consideration of this paper I infer that he considers the head of Red Mt. creek in that region, which he calls a crater-like depression, to be an extinct geyser-basin, and that the ore-deposits opened by the mines now worked there have been formed by geysers. Now a geyser, as its name implies, is a spring which violently ejects its waters into the atmosphere, and hence, even more than other springs, is essentially a surface-phenomenon.

As, since the ore-deposits of Red Mt. basin were formed, something like 2000 feet in thickness of rock-material have been eroded away to form the basin, it is evident that they could not have been formed by a geyser, nor can it be determined at the present day whether geysers ever existed there or not. Apparently Professor Comstock has thought to find a resemblance between certain rounded ridges of light-colored rock near these mines with the mounds formed by deposits from the springs in the Yellowstone Park. Instead, however, of being surface-deposits, they are simply portions of the andesitic country-rock from which acid waters have removed the basic constituents, perhaps depositing a certain amount of silica in their place; the resulting quartzose masses, offering greater resistance to the disintegrating effect of atmospheric agents and to erosion than the surrounding rocks, have been left as mound-like ridges protruding above the general surface, more or less independent of the natural drainage-channels.

There are abundant mounds formed by deposits from springs in the vicinity of Red Mountain, but they consist mainly of bog-iron, and are formed by oxidizing surface-waters passing through and decomposing bodies of sulphides of the metals, whose action is quite

* *Transactions*, vol. xv., 1886, pp. 218-265.

distinct from, and in some sense the reverse of, those which originally formed these bodies. It would seem almost unnecessary to insist on this distinction, but I recall the description given me of similar deposits in the Redwell Basin in the Gunnison region by Mr. J. K. Hallowell, who claimed as exact a knowledge of the geology of that district as Professor Comstock does of the San Juan region. According to him, galena and pyrite could be found in actual process of formation there at the present day. On personal examination, I found that a bog-iron deposit had been formed throughout the débris filling the bottom of this ancient glacier-basin, and in one place rested directly on the outcrop of a vein carrying pyrite, galena and other sulphurets. Owing to the extremely decomposed condition of the country-rock of this vein where the bog-iron rested on it, Mr. Hallowell had evidently concluded that it formed part of the spring-deposit, failing to recognize the dividing line between recent deposit and original, though decomposed, rock-formation.

Although, in studying a given deposit, it is not safe to assume that there has been a spring at that particular spot, from whose waters it was deposited, it is evident that the underground flow of waters which formed ore-deposits would have followed similar laws to those which regulate the flow of water now appearing at the surface as springs.

Natural water-channels.—From the study of the flow of spring-waters, three principal structural conditions may be conceived which would produce natural water-passages in the rocks which form the earth's crust:

1st. Sedimentation or bedding.
2d. Intrusion of eruptive masses.
3d. Dynamic movements producing fractures across rock-masses of differing origin or composition.

1st. From the first of the above causes, by the deposition of alternating strata of varying degrees of permeability, or by successive flows of igneous rocks, natural channels will be afforded parallel to the stratification or bedding-planes, and more or less coincident with them according to the nature of the material of which the bounding beds are formed. This is proved by the fact that where such strata or beds are inclined or plicated, and afterwards eroded so that their edges outcrop, springs will be found along such outcrops where they are at a lower level than the general mass of the strata, or artificial springs may be produced by artesian borings which

fulfil the necessary hydrostatic conditions. It is hardly necessary to quote instances of flow of spring-waters under the simple conditions of alternating strata of permeable and impermeable sedimentary rocks. The evidences of water-passage between flows of eruptive rock are well shown in Idaho and Eastern Oregon, where whole rivers disappear beneath the great lava-flows that cover such immense areas of surface, and reappear at some lower level at the bounding plane between the successive flows, evidently having sunk through some natural fracture and followed a more permeable bed of mud or breccia, such as is generally found alternating with the beds of solid lava.

These primary water-channels along bedding-planes may be interrupted by either of the two other causes mentioned above.

2d. Eruptive dikes or cross-cutting intrusive bodies of any form may interpose relatively impermeable masses across their course, or intrusive bodies, running parallel with the bedding, may render the plane of contact a more ready water-passage than it otherwise would have been. Thus in regions of eruptive activity, springs, especially thermal springs, are generally found along the outcrop of the contact of the later eruptions with the rocks through which they have passed.

3d. The interruption to these primary water-channels resulting from the varied forms of rock-fracture caused by dynamic movements are so manifold and numerous that it is not always possible to trace back cause from effect. Springs frequently issue from the outcrops of fault-planes; and, when crossed underground, the latter are generally found to be water-bearing. It is easy to conceive that where a series of beds, including certain permeable or water-bearing strata, are broken or displaced by fault-movements, the current along the permeable strata may be interrupted. On the other hand, the passage of water along this new water-channel may not be continuous to the surface, but may only follow it until it meets another permeable stratum and turn off again along that in a manner similar to the well-known occurrence in artesian wells, where, if the bore-hole be not protected by a water-tight casing, the water rising from some deeper bed may gradually be dissipated in other permeable beds passed through by the drill, between the water-bearing bed and the surface, and the surface-flow finally lost altogether.

It is evident that the greatest number of underground water-channels, though perhaps not those carrying the largest volumes of water, will be afforded by the multitudinous fractures in the crystal-

line or eruptive rocks and in the older and more metamorphosed sedimentary strata. In one sense these might be made to include the contact-channels along dikes and intrusive bodies of eruptive rock also; for it is probable that such bodies, when forced up from below, have followed some previously determined fracture or natural division-plane.

Before proceeding to consider how far the origin of various types of ore-deposits may be ascribed to the passage of mineral-bearing waters along one or more of the above natural water-channels, I will mention another possible class of division-planes in rocks which have been considered by some as the origin of mineral-bearing fissures, but to which, so far as I know, that of springs has never been ascribed. These are contraction-planes or joints. The most evident examples of them are naturally found in eruptive rocks, notably the prismatic joints of more recent eruptives. It is possible that the division-planes which tend to separate most eruptive rocks into parallelopipedic fragments may have been originally determined by contraction of the mass, but it is probable that the actual fracturing along these planes was produced by dynamic movements. Whether sedimentary rocks contract after deposition from the expulsion of the water they must have contained when forming the sea-bottom, may be considered an open question. One thing is evident: that contraction-planes must be confined to one rock-mass or bed, and cannot cross several of them, as do most mineral-bearing fissures. It is also evident that, as contraction-planes alone, there would have been no movement or pressure along them. Hence planes where evidences of movement or pressure are found cannot result from contraction alone. For these reasons, and the further one that I have never yet found ore-bodies deposited on planes which I could feel assured were the result of contraction alone, I think it safe to leave contraction-planes out of consideration, for the present at least.

Ore-deposits along bedding-planes.—The greater part of our ore-deposits are found in mountainous regions where eruptive and dynamic action has been most energetic; consequently, deposits resulting from the flow of water along bedding-planes alone, unconnected with the other classes of water-channels, form but a small proportion of the whole. There are, moreover, other conditions than the simple readiness of flow of water that come into play. It is apparent that deposition will take place more readily from a comparatively sluggish than from a rapid flow; hence conditions that

tend to retard the flow or cause a partial stagnation at a given point will favor precipitation at that point. Such would be the actual contact of an impervious stratum, *e.g.*, a bed of clayey material,—with a readily pervious one like a loose sandstone. Again, in plicated strata, points where by sharp folding the beds are more closely compressed together—as on the side of the fold—than at other points, are often found to carry larger bodies of ore. The chemical composition of the different beds is also a most important factor; so much so that in regions like the Ten-Mile district, where thin beds of limestone are found scattered through considerable thicknesses of sandstone and shales, the ore of the so-called bedded deposit is almost exclusively confined to the more readily attackable limestone. The term bedded vein or deposit, as derived from von Cotta's *Lagergang* (literally, *bed-vein*), might be objected to as seeming to imply that the ore was deposited contemporaneously with the enclosing beds, and for this reason the less concise term "deposit along bedding-planes" would be preferable. Where the deposit occurs at the junction of two beds of very dissimilar composition, like limestone and quartzite or slate, it is often called a contact-deposit, but it seems that this term should be confined to those of the next class, contact-deposits with rocks not of contemporaneous formation.

Instances of deposits of this class that occur to me are those of the Ten-Mile region, the Red Cliff region and others in Colorado, which are mostly along the bedding-planes between limestones and argillaceous beds, though at Red Cliff some are found in quartzites. The iron-ore beds in the Silurian rocks of the Appalachians are probably of this class, for Prime reports that he finds evidence that they are the replacement of the limestone rock, hence cannot have been formed, like bog-iron deposits, contemporaneously with the enclosing rocks, as Newberry maintains. They are not necessarily confined to the immediate bedding-plane, but in a porous rock may spread through the mass, like those of Silver Reef, where precipitation has been probably induced by the presence of some reducing agent of organic origin. The copper-deposits of Mansfeld in the Hartz, and the lead-deposits of Commern in the Eifel, may be found to belong to the same class, though they have generally been considered to be of contemporaneous origin with the enclosing Triassic beds.

It will be evident *a priori* that mineral currents following stratification-planes cannot be considered as necessarily coming from below, as has been assumed by many to be the universal direction of such

currents. As I have shown in a former paper,* the direction of flow of underground waters cannot be determined beforehand for a given point; as this flow is a circulation, its direction may vary according to the local conditions which would govern it. As far as my observations go, in deposits from currents following stratification-planes the waters have sunk into the bed from its upper surface as if under the influence of gravity.

Deposits along contact-planes.—Mineral-bearing solutions gathering in or flowing along the planes of contact of eruptive bodies with rocks through which they have been forced would have a tendency to deposit their contents along such planes, whatever the direction of their flow. If they were ascending currents, it may be conceived that they were coming from a hotter region, or from the vicinity of a larger and not so thoroughly cooled mass of igneous rock, where their solvent power was greater, to a cooler region in which this solvent power would be relatively less. If lateral or descending flows, gathering from the mass of one of the walls of the fissure, precipitation might be induced from the solutions thus brought in by a retardation or temporary stagnation of the flow, by dilution through mixing with other waters already circulating in the fissure, or by some chemical interchange resulting from contact with the other wall, if a rock of different chemical composition from that through which the solutions had been passing. For these reasons it seems advisable to distinguish such deposits from those along stratification-planes on the one hand or along rock-fractures on the other, even in those cases where the contact-plane may be in part, or wholly, coincident with a stratification- or fracture-plane.

It is quite a common practice among miners to designate as "contact-deposits" ore-bodies occurring along the dividing plane between two bodies or beds of different rocks, even when such plane is simply a stratification plane. Such practice should in my opinion be avoided, and the term confined to the planes defined above, which are distinctly of later formation than stratification-planes, and in which the conditions of deposition would have certain distinctive characters.

Contact-planes as defined above will more frequently be found to coincide or connect with planes of rock-fracture, since one can hardly conceive of sheets of eruptive rock being forced through existing rock-masses in the form in which we now find them, unless they had followed some already determined line of fracture, or at

* "Genesis of Certain Ore-Deposits," *Trans.* xv., 137.

least of readiness to be forced open, and it is well known that eruptions of lava at the present day are generally preceded by earthquake shocks, which probably involve a very considerable shattering and fissuring of the earth's crust in the vicinity of the eruption.

Ore-deposits along contact-planes are very common, and mining engineers can doubtless recall frequent instances which have come under their own observation. The famous Leadville deposits are in great part instances of the less common type of deposits, in contact with intrusive bodies which have generally followed stratification-planes. Among them, however, are deposits along the contact of cross-cutting eruptive sheets, the so-called second contacts, and in some instances the ore-currents have followed rock-fractures, and hence belong to the third class. As in all deposits in limestone, owing to the readily attackable character of the rock, the actual form of the various deposits at Leadville is extremely irregular; more so than it would be in a more siliceous rock; but I have as yet been unable to conceive of any classification, founded on the form alone, that would essentially aid in the description of such bodies, or help the miner in exploiting them.

Of deposits along the contacts of dikes or cross-cutting bodies of eruptive rock, hence generally occupying a more nearly vertical position, abundant examples are found throughout the Archæan areas of the Rocky Mountain region, most of which are commonly classed by the miner as fissure-veins, because of the prevailing prejudice in favor of the supposed greater value of that type of deposit, It may be that the whole mass of a narrow dyke is impregnated with mineral, and thus constitutes the ore-body; but in the structural sense it is none the less a contact-deposit, since the deposit has been made by waters acting from the contact-planes outward. Instances of such deposits are frequent in Boulder, Gilpin, and Clear Creek counties, in Colorado.

On the other hand when the deposit occurs along a plane of contact of two dissimilar bodies of rock brought into juxtaposition by faulting, like the Comstock lode, it would more properly belong to the third class.

Deposits along planes of rock-fracture or fissures produced by dynamic movements.—Many somewhat divergent views have been entertained at various times by geologists as to the origin, manner of formation, and even the proper designation of the many kinds of fissures, cracks and joints that traverse the rock-masses forming the earth's crust. Those that I propose to briefly present here are the

ones which a long field-experience has shown to accord best with the facts of nature, and which, as far as my reading has extended, are in essential harmony with those entertained at the present time by the best structural geologists both of this country and of Europe.

The most prominent and readily remarked of these rock-fractures are the great faults which have played so important a part in determining the orographic relief of our globe, a part second only in importance to that of the flexures or plications of the strata, to which, as I shall show, they are closely allied both in origin and manner of formation. The greatest of these faults often extend for miles in length, and the displacement of the opposed rock-masses on either side of the fault may amount to several thousand feet. Minor faults, or displacements which are found in infinite variety, especially in regions of great dynamic disturbance, may not produce any readily apparent effect upon the surface-features, and yet may be recognizable as determining the flow of springs, or be detected by the underground workings of mines. They have been most thoroughly studied in the workings of coal-mines, where the importance of careful underground mapping is most generally appreciated, since the determination of the direction and amount of throw of such faults has an actual money-value. Much has been written about methods of making such determinations, but more attention seems to have been given to reducing them to mathematical formulæ than to tracing their origin or connection with other earth-movements. In all these rock-fractures the evidence of a movement of displacement, as disclosed by the discrepancy or want of correspondence in the adjoining walls, is usually very prominent.

There are other and much more numerous rock-fractures, in which there is either no movement of displacement, or it is so slight as to be with difficulty detected. Among them certain classes are characterized by their frequency and their general parallelism in two or more co-ordinated directions, and at angles often approaching a right angle with each other. To these, owing to their apparent uniformity of character, the name of joints, or cross-joints (inasmuch as they are apt to cut across the strata) has generally been given by English-speaking geologists.

As regards the designation of these various forms of fissure or rock-fracture, from the very fact that there has been so little agreement as to their genetic relations, there has necessarily been a great want of uniformity in nomenclature. In different countries, and even in different districts in the same country, local terms have been

used to describe phenomena supposed to be peculiar to the locality, which have been perpetuated by geologists and have thus introduced into geological literature a greater diversity and variety of terms than is justified from a strictly genetic point of view.

Daubrée, in his comprehensive view of the phenomena attending the flow of underground waters, recognizing the hopelessness of assimilating all these local terms, proposes the general name of *Lithoclases* (rock-fractures) for all rock-fractures of whatever nature, and, as principal subdivisions uses, *Paraclases*, to designate those in which the movement of displacement is readily apparent (in other words what are generally called faults), and *Diaclases* to designate those that are generally known as joints and in which there is no perceptible movement of displacement, but which are characterized by a parallelism and uniformity of distribution, and traverse different rock-masses. To both of these he attributes a mechanical origin, in other words considers that they are the result of dynamic movements of the earth's crust. He further designates all the rock-fractures of feeble dimensions, which are confined to a given rock-mass and render it liable to break up into small fragments under various influences, as *Leptoclases* (small fractures), subdividing these further into *Synclases* (contraction-fractures) or those which, like prismatic joints, are produced by the contraction of the rock-mass, and *Piésoclases* (compression-fractures), those small fractures, without any apparent regularity, whose frequent striated surfaces show them to result from a movement of compression like the *diaclases*.

On the other hand, Heim, who is the best representative of the German ideas on structural geology, in his classic work on the mechanism of mountain-building,* simply divides the larger rock-fractures into two classes: fold-faults (*Faltenverwerfungen*) and fissure-faults (*Spaltenverwerfungen*), both of which are produced by tangential or lateral compression of the upper portions of the earth's crust. In the former, this pressure has resulted in a general plication of the beds involved, which has progressed to such a point that the limit of their plasticity or capability of bending has been passed, and the tension relieved by actual fracture and displacement. In the latter, the fracture has taken place without any perceptible bending of the rock-masses adjoining the fault, the tension being relieved by sudden shatterings in the nature of earthquake-shocks, or because the lines of fracture were already determined, and the cohesion suffi-

* *Untersuchungen uber den Mechanismus der Gebirgsbildung.* von Albert Heim. Basel, 1878. II. Band, p. 44.

cient to admit of the preliminary plication of the beds no longer existed. He afterwards traces the effects of the same forces upon the internal structure of individual rock-masses, and shows that cleavage and foliation planes are, so to speak, diminutive faults, and that the effects of these movements are appreciable even in the microscopic structure of the rocks.

While Daubrée's nomenclature, as such, is the most complete and comprehensive thus far presented, the attempt to introduce into so practical a science as mining geology so many new and strange terms would seem to be of very doubtful advisability. For the present purposes, therefore, it seems sufficient to use the terms *fold-faults*, *fissure-faults*, and *compression-joints* to designate the most marked types of the various rock-fractures, bearing in mind that, as they are supposed to be the result of the same general force of compression, it may not always be possible to draw a definite line of distinction between the types.

Causes of fracture.—It seems hardly necessary to state at the present day that the force which produces the folding, faultings and rock-fractures, in short the mountain-building force, is to be considered as a result of the secular contraction of the earth's crust. There have been of late years, however, some theoretical arguments presented by physicists tending to throw doubt upon the adequacy of the force which might be produced by secular contraction to account for what they assume to be observed facts. Further, certain American geologists have thought to find conditions which, according to their reading of the facts, contraction could not account for, and for some of which it has even been thought necessary to revive the old and long ago abandoned hypothesis of a vertical upthrust.

As regards the theoretical objections it may be said, that mathematical demonstrations are of questionable value in a science like the geology of the present day, in which the exactly determined facts are as yet too few to afford premises of mathematical exactitude upon which to base them, and that those who are arguing against the contraction-theory have presented no adequate hypothesis to take its place.

On the other hand, among practical observers in geology the personal equation forms so large a factor that it may be fairly questioned whether the readiness of the gentlemen referred to above to find facts that could not be accounted for by the contraction-theory was not enhanced by a previously formed opinion of its inadequacy and a

desire to see it disproved. In my own pretty large experience in the same general field in which they have worked, I have found that this theory not only accounts for, but is the only one that *will* account for, all the observed facts: and that, where my observations extended over regions actually mentioned by them, I have found their explanations not to be in accordance with the facts of nature, as I read them.

Under the contraction-hypothesis, the forces exerted may be concisely described as resulting from the attempt of an already consolidated crust to fit itself more closely to a shrinking nucleus. Their effect is felt probably only upon a comparatively thin outer portion of the earth's crust: at any rate it is a very thin portion of this crust which comes under our observation. This crust may be conceived, therefore, as having been since its first formation in a condition of tension, a gradually increasing force which from time to time found its relief in earth-movements producing corrugations on its surface, and hence relative elevations along certain orographic lines, which from some reason or other were lines of least resistance or of weakness. Such lines, once determined, have been the scene of most marked expression of these constantly recurring movements of relief from tension. Closely connected with such movements have been the eruptions of igneous material, forced up from below, either from a region of permanent fusion of the material of the earth's crust, as was formerly most generally maintained, or, acording to later views, from local reservoirs brought into a fused condition as a more or less direct result of these movements, by the disturbance of equilibrium between the various forces involved in the general condition of tension. Whatever their source, the eruptions of igneous rocks have unquestionably had a very close connection with orographic disturbances, and further have indirectly played an important part in the formation of most ore-deposits.

Observation teaches us that these successive periods of dynamic disturbance, or of mountain-building, must have been followed by periods of relative quiescence, during which the regions elevated into land-masses were worn down by atmospheric abrasion, and their comminuted débris carried into the adjoining oceans to form a new series of sedimentary beds. Each successive series of dynamic movements would involve not only this newer series, but also the older and already plicated and fractured series of rocks; and thus the structural conditions are found to be the more complicated and more

difficult to decipher, the older the rocks in which we have occasion to study them.

The rock-masses of which the earth's crust is formed possess to a certain degree both rigidity and plasticity: the relative degrees of these qualities may vary not only with the composition, association and molecular structure of the different masses, but, in the same mass, with differing conditions of heat and pressure under which it exists at a given moment. Hence it is impossible to determine *a priori* whether a given amount of compression will produce plication and fracture combined, or fracture alone.

In a general way we know that certain rocks are more plastic than others: for instance, argillaceous rocks more than siliceous rocks; amorphous rocks more than crystalline rocks. Further, it is evident that under moderate temperatures distinctly stratified rocks will be more readily plicated than massive or crystalline rocks; in other words, under given conditions the latter would be more fractured than the former.

The fracturing of rocks, moreover, takes place the more readily the lighter the load upon them; in other words, plasticity increases with pressure. Thus, at sufficient depths below the surface, or under the weight of a sufficient mass of superincumbent rocks, fracturing of the rock-mass would no longer be possible, and compression would only result in some plastic deformation. Heim places at 15,000 feet the depth at which all fissures sufficiently open to admit the passage of waters must necessarily cease.

Heim further shows that the plasticity of a given rock-mass will be greater under a slow working pressure than under a sudden shock, or under the effects of a force of rapidly developed energy. Now it is a fact of observation that the elevation of mountains, which is in the main the result of plication, is an immensely slow movement, lasting at times through entire geological periods. We can readily conceive, therefore, that while, as observation shows us, under this slow-moving force great thicknesses of beds have been folded together like so many sheets of paper, when the limit of plasticity was reached and fracture took place, that this would be a relatively rapid action, more in the nature of sudden shock; indeed, that under such enormous pressure as must necessarily exist, the movement past each other of the lips of the fracture, involving hundreds or even thousands of feet of rock-masses, may have produced a series of violent shocks, which propagated themselves through the adjoining rock-masses, and produced in them a series of fractures

such as we see in so-called cross-joints. According to the theory that earthquakes are the relief of tension in the earth's crust, this propagation of movement would resemble the vibratory movement observed in modern earthquakes. Such an origin for cross-joints has already been suggested by W. O. Crosby.*

It has been objected, however, that while the propagation of such vibratory movements would produce the main parallel joints at right angles to the direction of movement of the wave, it is not so easy to account for the correlated joints which cross the primary joints, generally at a large angle.

If, however, we assume that all the rocks thus fractured were under a tension which, acting more slowly, would have produced folds, and if this tension was not simple but complex, or acting in more than one direction at the same time, so that not one, but two or more systems of folds would finally be produced; then, by the crossing of these systems of folds, there would result not only a plication along parallel axes, but at the same time a torsion of the beds or rock-masses involved. The assumption of a torsional strain, combined with that tending to produce plication, admits of an almost infinite complexity of rock-fractures as a product of these forces under the various conceivable conditions attending dynamic movements.

The foregoing theoretical consideration of the forces which may have produced the various rock-fractures under consideration has been purposely made very brief, and is hence possibly somewhat incomplete, for the reason that in so practical a science as mining geology, the outward manifestations of geological phenomena—the effects—are, in my opinion, of more importance than the causes, which always remain to a certain extent in the region of hypothetical conjecture, and that if one becomes too much wedded to a certain theory he is in danger of adjusting his facts to that theory.

I shall, therefore, insist more particularly upon the physical phenomena which characterize these rock-fractures, as determined by actual observation, and for which, while the above-mentioned hypotheses seem at present to account for them, I shall be quite ready in the future to adopt any other explanation that, by my own studies or by those of others, may be presented as more reasonable and adequate.

Common characteristics of compression-fractures.—There are three phases of structural evidence of rock-fractures and displacement

* *Proc. Boston Soc. Nat. Hist.*, vol. xxii., pp. 72-85.

resulting from compression, one or more of which I have found to characterize the various types of fissures carrying ore-deposits, which have come under my observation. These are:

First. Striations and "slickenside"-surfaces.

Second. Breccia or fragmentary material in the fissure itself, or zones of crushed or broken rock-material included between intersecting systems of fissures.

Third. A sheeting of the country-rock parallel with the main fracture: in other words, the occurrence of a system of minor fractures which divide the country-rock up into a system of approximately parallel plates or sheets. The distance between these parallel fractures, or the thickness of the sheets, may be reckoned by inches, by feet, or by hundreds of feet, according to the varying texture of the rock-masses involved, or the different dynamic conditions which have produced the fracture.

It will at once suggest itself that these are all phenomena characteristic of faults. But they are also found, at times, where there may be no recognizable evidence of actual displacement of the rock-masses on either side of the fissure or fracture. On the other hand, it will be equally evident that fissures characterized by these phenomena can hardly be the result of contraction, or shrinkage-cracks.

Striations are not confined to well-defined fissures, but are found on smaller planes within rock-masses; but in any case they seem to necessarily give evidence of movement under pressure, be the amount of that movement ever so small.

Fragments of country-rock are often rounded, and writers upon ore-deposits are accustomed to speak of them as having fallen into the fissures from the walls, and, when rounded, as having become so by attrition either against the walls or against each other. As regards the falling in, which seems to imply a fall in a free or open space of considerable dimensions, my observations have led me to consider it of rare occurrence, and to infer that the fragments generally found have been produced rather by the rubbing or dragging of one wall against the other. The greater or lesser size of the fragments would, in a measure, prove a greater or less distance between the walls, but it seems that under the enormous pressure that must have accompanied these rock-fractures, the space between the walls must have been more or less completely filled with attrition-material, only part of which would be actual rock-fragments, and the rest finely comminuted material, which, under the dissolving

agency of percolating waters, would finally result in more or less impure clays. The rounding of the fragments, on the other hand, is readily accounted for as the action of these same percolating waters, it being a well-recognized fact that the decomposing action of moisture in any form acts more rapidly on the corners or angles of a rock-mass than on its flat surfaces, and the sharper the corner the more rapidly is it eaten away.

Crushed zones are merely larger phases of the same actions as produce the breccia-material, and are subject to the same general laws, only differing in their greater dimensions and the more irregular shape of the enclosing walls.

The sheeting of the country-rock in faulted or fractured regions where ore-deposits abound, is a phenomenon to which hitherto too little attention has been paid. Its importance as a feature of fissure-veins is, however, great both from a geological and from a practical point of view. That it has hitherto escaped due recognition, is probably due to the prevalence of the old idea that vein-deposits are necessarily the filling of open fissures, and to the failure to appreciate to how great an extent they are actually the replacement of rock-material rendered more readily accessible and attackable by the dynamic movements which produced the fissure. This feature will be more fully illustrated in the practical examples given later.

Before proceeding to the consideration of how far these features appear in actual ore-deposits, it will be well to examine the typical manifestations of the above-enumerated rock-fractures where their relations are best shown, independently of whether they may have been mineralized or not.

Typical rock-fractures.—The typical form of the *fold-fault*, as observed by Heim in his studies in the Alps, and as shown experimentally by Daubrée in his "Géologie Expérimentale," may be produced in the following way: If a given series of strata by longitudinal pressure are compressed into a sharp anticlinal fold, the individual strata will tend to expand or swell along the axis, or in the crest of the anticline and in the bottom of the adjoining syncline, while in the intermediate portions—the sides, or as the Germans express it, the shanks (*Schenkeln*) of the fold—they will be contracted and attenuated or drawn out, as it were. When, therefore, the beds in folding have reached their limits of plasticity, the fracturing will take place by preference along the contracted portion ; that is, on the sides of the fold rather than along its axis. Such a fault, it is evident, will be necessarily accompanied by marked plication of the strata on either

side. Faults exist in nature, however, in which there is no marked plication of the strata on either side, but which are quite evidently the result of folding, since at one or both ends they pass into an unfractured fold. It is evident, therefore, that the latter type may be undistinguishable from the fissure-fault, in cases where the structure of the adjoining region is so obscured that it is not possible to determine whether the fault actually passes into a fold or not.

Characteristic examples of the above faults occur in the Mosquito Range near Leadville, Colorado, and will be found illustrated in the maps and sections of the Atlas accompanying Monograph XII. of the U. S. Geological Survey. There the Mosquito fault, which splits into two at its southern end, each branch terminating in a fold, and to the north has been traced for a distance of about 25 miles and may extend as much farther, is a typical illustration of Heim's definition of the fault-fold. Where the adjoining strata are exposed and have not been eroded away, they are seen to be sharply folded and to have once formed part of an S-fold, which has been fractured along one side of the anticlinal axis or on the shank of the fold. The Iron and Carbonate faults, on the other hand, which, though of less extent, it has been possible to study more thoroughly, owing to the many mine-openings along their course, show no evidence of folding in the bending of the strata immediately adjoining them on either side, which are cut across by the fracture nearly at right angles to the bedding-planes throughout the greater part of their extent. They form, however, a definite part of a system of gentle anticlinal and synclinal folds, and pass into an unfractured fold at either end, being, therefore, properly speaking, fold-faults.

In other prominent faults in the district, such as the Ball Mt., Mike, and Pilot faults, it is not possible to trace their direct connection with folds, and yet it is evident that they were produced by the same general dynamic movements as the others, and form part of the same tendency toward plication, which, in their case, has evidently been counteracted by the greater rigidity of the rock-masses as a whole, resulting from the large proportion of igneous masses entering into their composition.

The phenomena of striation, brecciation and sheeting of the country-rock are well seen in the Iron and Carbonate faults, which are the only ones that have been explored underground. As I have shown elsewhere,* these fault-planes are not mineral-bearing, since the

* "Notes on some Colorado ore-deposits," *Proc. Colo. Sci. Soc.*, vol. ii., Part II., October, 1886, p. 85.

principal mineral deposition in the region was accomplished before the faulting took place. But in cases where the faults cut through ore-bodies, considerable broken and ground-up vein-material has been dragged into the fault-fissure, in quantity sufficient to give rich returns for extraction. The brecciated character of this material is, however, less readily recognized than when the fragments have been cemented together by metallic minerals, as is the case in many fissure-veins. The sheeting of the country-rock in these two faults has apparently been confined to the side of the fault opposite to that from which the pressure came, and has produced what are sometimes called step-faults, the movement of displacement having been partially distributed on several minor planes, a hundred feet or more apart, and parallel to the main fault-plane.

The best region for studying the typical *fissure-fault* that has come under my observation is the southwestern portion of the Elk Mountains, in Gunnison County, Colorado. This has been a region of intense and repeated dynamic disturbance, accompanied by enormous eruptions of a great quantity of igneous rocks in all the varied forms in which they occur. Elevated glacial amphitheaters or basins, at altitudes of about 11,000 feet, whose walls consist of thinly stratified and nearly horizontal beds of variegated colors, afford unusually favorable opportunities for tracing the many faults, which form a complicated network over a large area, and for determining the amount of displacement produced by them, which is generally slight and measured by tens or hundreds of feet. Although, as seen on a cliff-face, these faults seem to consist of a single plane of fracture, it is found, when they are examined in underground workings, that what might be called the fissure-plane consists of a series of thin sheets of more or less altered and intensely compressed and striated country-rock, generally only an inch or two thick. Where, as is frequently the case, these fissures have been mineralized and constitute important ore-deposits, the original character of the sheets is not readily apparent; but the amount of breccia, consisting ordinarily of small angular fragments of country-rock, cemented by vein-material, sufficiently proves that the fissure is a fault-fissure; and, when explorations on the strike have extended beyond the zone of mineralization, the sheets of country-rock are found to retain enough of the original structure to prove their origin. Further, when cross-cut drifts have been run into the country on either side, a series of more or less parallel cracks or fissures are found, gradually disappearing as the distance from the central fissure increases.

As only the ore-bearing portions of such fissures are, as a rule, explored, it is evident that the fracturing and the directions and extent of the fissures can be determined but incompletely. Still, owing to the frequent surface-exposures, a certain regularity of direction is easily detected, showing that the fractures have been determined by definite series of dynamic movements. Although in this particular region the faults are unaccompanied by any considerable plication, yet at no great distance, and nearer a center of disturbance, there has been intense plication, accompanied by fold-faulting.

Similar fissure-faults, with the same phenomena of thin sheeting of the country-rock, and with breccia cemented by vein-material, are found in the San Juan region. Here, also, the directions of the various faults follow certain systems, which a thorough orographic study of the region would, doubtless, enable one to connect with the dynamic movements which produced them.

From these individual fissure-faults there is a gradual transition into the co-ordinated fractures, as a rule greater in number in a given area but of less individual extent, which form a sort of fractured zone with two or more prominent directions of fracture, apparently of nearly contemporaneous formation, which I have called *compression-joints*, because where I have studied them in mining districts, either by personal observation or from the description of others, I find one or more of the evidences of compression present, viz., striation, brecciation or crushed material, and sheeting of the country-rock. Such complicated systems of fracture would appear to involve, as I have already suggested, the action of more than one system of dynamic movement; that is, of forces of compression acting at the same time in several directions, and hence combining with the direct plicational strain more or less strain of torsion.

The effects of such torsional strain can be best understood by considering the effect of torsion alone. This is well shown in Daubrée's experiment upon a sheet of glass held firmly at either end, and subjected to a torsional strain up to the point of rupture. The following diagram, Fig. 1, showing the cracks thus produced in the glass, is copied from that in his "Géologie Expèrimentale."

It is to be remarked in regard to the cracks or fissures thus produced:

First. That they follow two general directions, crossing each other at a nearly uniform angle.

Second. That certain cracks are more prominent and fully devel-

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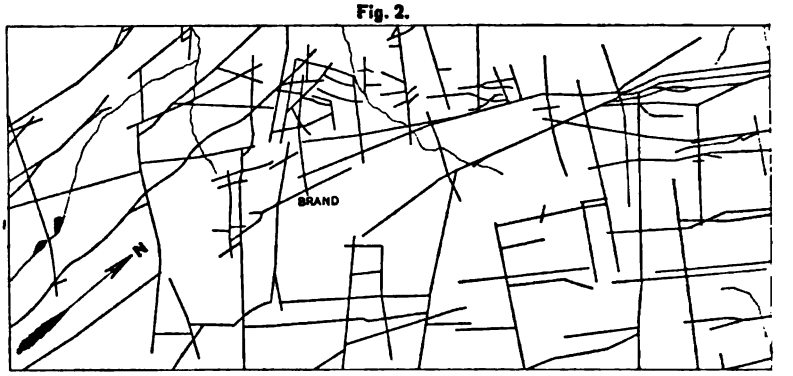
veins were the filling of considerable open fissures by mineral currents coming from some unknown source below, and has been rather mineralogical than structural; that is, more attention has been given to the character of the filling than to the structural origin of the fissure. If all those who have described mines and ore-de-

Fig. 2.



Veins in the Erzgebirge, near Freiberg, Saxony.

posits had devoted as much and as intelligent study to the structural features of the region where they occur as the late von Groddeck did in the Hartz, we should probably be much nearer a satisfactory theory of the origin and manner of formation of the ore-deposits than we are at the present day.

It is evident that by a succession of dynamic movements, especially when accompanied by torsional strains, an almost infinite variety of fissures and passages for mineral-bearing waters may be produced, and that it would therefore not be possible beforehand to describe all the various structural conditions under which ore-deposits may occur. It is only by an intelligent description of such fissure-deposits as are observed in nature in all parts of the world that we can get a comprehensive view of the various possibilties. Still there are certain conditions that suggest themselves as a result of the structural method of considering them that would seem to have a general application.

Structural generalizations.—Extent of fissures.—Since the dynamic movements are confined to the crust of the earth, it is evident that the fissures produced by them cannot literally have an indefinite extent in depth, though in certain cases it is very possible that this extent may be practically indefinite, as it may go beyond the limits at which mining is practicable. It is fair to assume that

those fissures which have the greatest horizontal extent will have the greatest extent in depth; in other words, that their vertical and horizontal dimensions bear some sort of proportion to each other. If, therefore, as some have maintained, the vein-filling has in all cases been brought from some source at great depths in the earth, the greatest fault-fissures would be expected to be the greatest and most frequent ore-producers, since they would reach nearer to this unknown source.

In my own experience, however, I have found rather the reverse to be the case, which, as far as it goes, furnishes an argument in favor of the view that the vein-material has been derived from the surrounding, though not of necessity absolutely contiguous, rocks. On the great fold-faults I have found no considerable deposits of ore, and it is comparatively rare that continuous deposits are found along a single well-defined fault-fissure. The majority of deposits seem to occur where there are a series of fissures, more or less regularly co-ordinated, in which several of the series are prominently accentuated. In such systems there seems to be a tendency for the rich ore-bodies or bonanzas to extend in a direction which lies at an angle with that of the main fissure, or to continue for a certain distance along one fissure and then to pass into another fissure, set off at a little distance from the first. It would seem probable that there must be some structural reason for the concentration of ore in this way, and that sufficiently wide and detailed studies might discover this reason and thus throw some light on this, at present, so obscure subject.

The practical lesson to be learned from the above phenomena is that the miner should not confine his explorations to the single fissure in which his ore occurs, but when he runs out of bonanza in that, he should seek a continuance of it in some adjoining fissure or plane, in a direction to be determined by the study of the system of the fracturing of the region and of the general direction of the bonanzas.

Vein-walls.—The second generalization is in regard to the *walls*, which have generally been considered an important and almost indispensable characteristic of a true fissure-vein. The typical wall which the miner considers an evidence of a strong and well-defined fissure-vein is a comparatively smooth, generally striated, rock-plane, and frequently coated with a clay selvage—a band of decomposed argillaceous material which itself generally shows evidence of pressure and movement. From the above structural point of view of the origin

of vein-fissures it is evident that the character of the wall and selvage is dependent on the composition of the rock and the amount of displacement and pressure. The grinding of one face of rock against another will undoubtedly tend to plane both off and to produce a certain amount of fine attrition-material; but this attrition-material will not necessarily be reduced to clay unless it has further been subjected to the decomposing action of water, which has carried off certain portions and left an argillaceous residue. The extreme instance of such decomposition is found in the muddy accumulations at the bottom of caves in limestone, which are simply the less soluble residues, mostly silica and alumina, resulting from the dissolution of large quantities of more or less impure limestone.

These walls and selvages are a frequent accompaniment, but by no means an essential characteristic, of an ore-bearing fissure. It is quite conceivable that one or both may be wanting; and such occurrences are not uncommon in nature. Take, for instance, the veins of Butte, of which I gave a brief description at a former meeting.* These are a series of co-ordinated fractures or compression-fissures in a remarkably homogenous mass of granite. Apparently there has been little or no displacement of the walls of these fissures relatively to each other; hence, but little attrition-material has been produced; and for this reason—and probably, also, on account of the character of the rock and because it was not much decomposed along the fissure-planes before the advent of the ore-bearing solutions—no clay selvages have been formed, and the ore-bearing solutions have eaten into the wall-rock to varying distances, replacing it more or less completely by vein-material, and leaving no definite boundaries or walls to the deposits. There is no reason, however, for considering them any the less true fissure-veins or less valuable ore-deposits on this account.

On the other hand, under certain conditions, instead of an absence of well-defined walls there may be so many as to mislead the miner who depends too implicitly upon them as a boundary of his ore-body. In the Gunnison region above mentioned, for instance, where, owing to the plasticity of the country-rock, it has been divided along the main fracture-planes into a series of very thin parallel sheets, the space between these sheets has frequently been filled by quartz, which thus forms a thin sheet, often so completely reproducing the form of the fissure as to present a cast of the striation-

* "Notes on the Geology of Butte, Montana," *Trans.*, xvi., 49, July, 1887.

surfaces. Such a sheet of quartz, when the adjoining bands of country-rock have been replaced by vein-material, forms a hard, well-defined wall to the ore-body, which delights the eye of the honest miner and enhances in his mind the value of his property.

Ore may be found as well on one side as on the other of such a wall, and not unfrequently is apparently confined to one side for a considerable extent along the length of the vein, and then is found almost as exclusively on the other side. Thus, in one prominent mine in this district I found a drift run for several hundred feet in barren country, but following what was apparently the direct continuation of the vein which had been yielding a large amount of rich ore. After a change of administration in the mine, it was found by cross-cutting that there was a continuous ore-body only 4 to 6 feet to one side of this drift and parallel with it. In another mine in the same district, which had produced a great deal of very rich ore, I found that though there were frequent crosscuts into the country on the hanging-wall side, disclosing the usual systems of parallel fissures, none had been made on the foot-wall side. The reason given was that this foot-wall was so uniform and well-defined throughout the mine that it was considered useless to explore beyond, since it must necessarily be the limit of ore in that direction. An examination showed that there was no geological reason for this assumption in the character of the rock, and that it was simply one of a number of quartz-fillings between two sheets of country-rock. On visiting the mine a few weeks later, I was told that in the southern portion of the mine, where the vein had seemed to be running out at the time of my first visit, a new body of rich ore had been struck by cross-cutting into the foot-wall country.

The moral is that judicious cross-cutting forms a very important part of vein-mining, but should be conducted with due regard to the fracture-system of the adjoining country, and to the evidence to be obtained as to the course followed by the ore-bearing currents, or it may involve an unnecessary amount of dead work.

Banded structure.—In most of the deposits of the Gunnison region, referred to above, there is a noteworthy appearance of banded structure parallel with the walls of the fissure. The evidence of faulting and of the thin sheeting of the country-rock is there so clear that the explanation at once presents itself that this appearance arises from the fact that the deposits are partly a filling-in of interstitial spaces, and partly a replacement of thin sheets of country-rock, the differing composition of the bands resulting rather from the necessary variation in

the process of deposition than from essential differences in the ore-bearing solutions. Were one to examine there only a large body of rich ore, and neglect to examine the adjoining poorer deposits and to study the structural conditions of the region, one might be led to adopt some of the complicated explanations set forth in books on ore-deposits, such as successive reopenings of the vein, to account for the conditions found, instead of the simple one given above. Reasoning inversely, I am led to think that much of the banded structure described in books on ore-deposits might be accounted for in this way, if the deposits could be re-examined with a view to determining the structural evidence in favor of it. For instance, in the excellent work of J. Arthur Phillips* there is given a diagrammatic section of a vein at Carn Marth, in Cornwall, consisting of six successive bands composed of quartz, with metallic minerals in some of them, separated from each other and from the adjoining country-rock by clay partings or selvages. The explanation there given to account for this condition of things is, that it is "a fissure that has been several times reopened;" and the author says, "It will be observed that each reopening has been attended by an amount of grinding action between the walls sufficient to produce a clay parting of considerable thickness." This explanation involves so much that is geologically improbable that I feel convinced that it is not the true one, and that some condition of things analogous to that described above might be found on examination of the mine itself. The clay parting might prove to be a more or less completely decomposed band of country-rock; for it is difficult to conceive that clay could be produced by the grinding of quartz on quartz. In reading such descriptions one has to bear in mind that in making a diagrammatic section one is liable, consciously or unconsciously, to bring out strongly points that are in favor of the explanation one has in mind, and to neglect those that are opposed to it. That there may be repeated or continued movements along the same fracture-plane cannot be denied, but, as will be argued later, one should not resort to such explanations for phenomena that can be accounted for otherwise.

Crushed zones.—Thus far we have been considering mainly the sheet-like bodies deposited along fissures having a generally parallel direction. But cases must also occur where the systems of rock-fracture intersect each other, and under suitable conditions consider-

* *A Treatise on Ore-Deposits.* London. 1884, page 48.

able portions of the country-rock included between such intersecting fractures may be broken up or crushed to such an extent as to admit a relatively free passage of percolating waters. Where such waters are mineral-bearing, the interstices between the fragments will offer spaces for the precipitation of their contents, or where the rock is readily soluble the fragments themselves may be replaced by vein-material. The forms of ore-bodies thus deposited may evidently present much greater variety and irregularity than those deposited along single or parallel fissure-planes. Mr. J. S. Curtis, in his memoir on the Eureka district,* gives a detailed description of the extremely complicated and irregular zones of crushed limestone to which the ore-deposits there are practically confined.

Where three or more nearly vertical fracture-planes intersect each other near the same point the prismatic body of rock included between these intersections may be so crushed and broken as, in a district rich in mineral-bearing solutions, to give rise to what are generally known as chimney-deposits. Where the fracture-planes are merely joints along which there has been but little movement, and no clay selvages have consequently been formed, the ore-solutions will eat out into the rock in such a way that the ore-chimney may appear to have a rounded instead of an angular horizontal section, and the fracture-planes themselves may have become, by the decomposition of the adjoining country-rock, so obscured as to be with difficulty traced in the immediate vicinity of the ore-body. Instances of deposits to which this origin may be ascribed are the Bull-Domingo and Bassick mines in the Silver Cliff district, and the Yankee Girl mine in the San Juan region of Colorado.

The first-named deposit occurs in Archæan rocks, traversed by a dike of compact syenite which in places forms one boundary of the ore-body. The ore is mainly galena, which constitutes the cementing material of fragments of the country-rock resulting from the movement of displacement. These fragments are mostly of gneiss, and as this rock is readily decomposed by aqueous solutions, the fragments are generally rounded; the few fragments of the less readily attackable syenite, however, retain in great degree their original angular form.

The deposit of the Bassick mine is a nearly vertical chimney of somewhat square or lozenge-shaped outline, which has been explored to a depth of about 1100 feet from the surface. It occurs in an

* *Monograph VII., U. S. Geological Survey.* Washington. 1884.

andesitic breccia, composed of angular fragments of various sizes and forms cemented together by material of essentially the same composition as the fragments. This cementing material, probably by reason of its later consolidation, has been more readily attackable than the fragments, for the ore has replaced it, forming concentric layers around the latter, whose angles have become rounded during the process of deposition. Here the fragments were original and not necessarily produced by the dynamic movements, which probably resulted in a simple fracturing of the rock without much displacement.

In the Yankee Girl there are several chimneys in which the ore occurs. They are of elliptical outline, the longer axis corresponding in direction with a main system of fractures running through the region. Although the striated surfaces of these planes show that there has been movement along them, there is but little evidence left of actual brecciation of the country-rock, the ore-solutions having completely replaced the andesitic country-rock between the fracture-planes which gave access to them: in places a siliceous skeleton is left, the basic constituents being replaced by vein-materials; in other places a solid body of metallic minerals is found, while the country-rock adjoining the body of pay-ore is impregnated to a considerable distance with low-grade sulphurets.

Repeated movements along fissure-planes.—It is a well-recognized fact of structural geology that in successive dynamic movements in the same region there will be a tendency for fractures to follow already determined planes of fracture. Furthermore, it appears that the faulting of a rock-mass is not necessarily a geologically instantaneous movement, but that the displacement may continue for some time after the first fracture has been determined, probably dying out very gradually. Some such continued movement seems necessary to give time for the reduction by the action of water of the attrition material to the clayey condition in which it is often found. We may expect, therefore, to find evidence in large fissures of repeated movements. In the Comstock lode, for instance, which Mr. G. F. Becker has so well demonstrated* to be a great fault-fissure, accompanied by sheeting of the country-rock on a grand scale, the crushed and sugary condition of the quartz, which in the fissure has replaced the included masses of country-rock, can hardly be explained, except by movement along the fault-plane subsequent to

* *Geology of the Comstock Lode, Monograph III., U. S. Geological Survey.*

this replacement, though perhaps prior to the final completion of the ore deposition.

On the other hand it is easy to conceive that the healing of a fissure, as we might regard the filling in of its interstices by vein-material, might make it better able to resist fracture than it was before, and a second fracture in such a case would not necessarily follow the already determined plane. Moreover, if we recall the diagram given above, showing the fractures produced by torsion in a sheet of glass, it will be seen that cross-fractures, and even apparent displacement of one fracture by another, may be produced by one and the same strain. Hence, in studying a system of fissures, one must not too hastily conclude that each direction of fracture means a distinct movement or even that displacement of one fissure by another necessarily proves that the latter was produced by a distinct and later movement; to be sure of this, some of the other evidences of movement must be found.

The famous Ontario mine in Utah affords an excellent instance of a strong fault-fissure deposit in a region which has been subjected to repeated dynamic movements, associated with successive intrusions of eruptive rock, and illustrates the above-mentioned points. I had intended to submit detailed plans of the underground workings of this mine, drawn to scale with the geological data indicated thereon, but find that the notes gathered somewhat hastily during my visit of last summer are not sufficiently complete for this purpose. I shall, therefore, describe, as clearly as I can in words, the characteristic structural features of the deposit.

It occurs in distinctly bedded quartzites of middle Carboniferous age, which dip about 20° to the northward and strike here nearly east and west. These rocks are traversed by numerous bodies of eruptive porphyry, some in the form of dikes, others of irregular shapes. But few appear at the surface in the immediate neighborhood of the mines; they are mostly disclosed by the underground workings, many of the dikes having a parallel direction with the vein-fissure and sometimes form one of its walls for considerable distances. An older dike is faulted by the vein-fissure.

The vein-fissure has a somewhat curving direction, running about east and west, or parallel with the strike of the formation in the eastern part of its course, and bending towards the southwest in the west half. Its dip, however, is much steeper than that of the quartzites, being 45° to 50° north, instead of 20°, so that it cuts these at an angle of about 25° to 30°. In the western part of the mine it is

a double fissure, or, as the miners express it, there are two veins, a north and a south one. These are connected by what the miners call "cross-courses," which are smaller, co-ordinated and nearly parallel fractures, forming an acute angle—about 30°—with the main fissures. These are regarded there, in accordance with generally received ideas, as a distinct system of fractures formed at a different time from the main fissure. I find no geological evidence of this, however, either in the structural conditions or in the composition of the vein-material.

The fissures show plentiful evidence of movement in striations and crushed country-rock, the ore being a filling-in of the interstices between the quartzite fragments, and perhaps a partial replacement of the same. The principal fissure consists in places of a zone of broken quartzite, sometimes over one hundred feet wide, the ore being distributed in seams along the boundaries of this zone and crossing it along a somewhat irregular series of planes, but in which a certain co-ordination of system can be traced.

Three distinct series of dynamic movements can be traced in the structural condition of this mine. The first is proved by the existence of a narrow dike of older porphyry, crossing at right-angles the western part of the vein-fissures, where the two members are about six hundred feet apart. This is locally called the trap dike from its somewhat darker color than the other porphyry bodies. By the second series of movements the vein-fissures themselves were produced and this trap dike was faulted and displaced. On the six hundred foot or drain-level of the mine this displacement can be observed in each of the principal fractures. The lateral or horizontal movement is about forty feet, and, as the striation-surfaces across the face of the dike are inclined at an angle of about 45° with the horizon, it may be assumed that the vertical movement was of the same amount. The third dynamic movement is shown by what is locally called the "dislocating fissure," which runs in the same general direction with the western part of the vein, and on the drain-level is intermediate between the two veins, cutting and displacing the cross-fissures.

There are also large faults in the country-rock at both ends of the workings, which now have a linear extent along the vein-fissure of about six thousand feet. I had not time to determine definitely to which period of movement these last faults belong.

A careful and accurate study of the structural relations of this most important mine, of which the above is only a hasty glimpse,

gathered during a couple of visits to a part only of the underground workings, would form a most valuable contribution to our knowledge of vein-structure.

Another mine recently visited by me, whose structural conditions seem peculiarly instructive, is the Queen of the West mine, near Kokomo, in the Ten-Mile district of Colorado. It is situated on the steep southeast face of a shoulder of Jacque Mountain, and the vein-fissure runs nearly parallel with this slope, or in a northeasterly direction, and has been explored to a length of about two thousand feet. It stands nearly vertical, thus crossing about at right-angles the sedimentary beds in which it occurs, and which have a slight dip eastward or down the slope of the hill. These beds consist of sandstone, rather rich in feldspar, some shaly beds, and intrusive sheets of porphyry generally conformable with the stratification of the sedimentary beds.

The faulting of these rocks has taken place not along a single plane, but along a series of parallel and closely contiguous planes. In other words, by the dynamic movement the country-rocks have been divided into thin sheets, each of which has moved past the other a certain distance, and in the central part of the fissured zone the interstices between the sheets have been filled with vein-material and the sheets themselves decomposed, impregnated, and to a greater or less extent replaced by it. There results a condition of things which is extremely puzzling for the miner who expects to find his ore between well-defined walls. Walls there are in abundance, and of excellent definition at times; but no one wall can be followed continuously for any great distance. Experience has, therefore, taught those who are managing the mine, while they follow in their main drifts or levels as nearly as possible the center of the mineralized zone or vein, to run frequent and extended cross-cuts on both sides of this central drift, which have disclosed ore-bodies running parallel with it, now on one side, now on the other, and often 15 to 25 feet distant from it. The longer cross-cut drifts, which have been run 30, 50, or even 100 feet into the adjoining country-rock, disclose a series of parallel fissures, in general farther apart as the distance from the central drift increases. Although mostly barren of pay-ore, they always give evidence of a certain amount of mineralization: sometimes they are filled by a vein of massive crystalline calc spar from a few inches to a foot in thickness. In the portion above the water-level, owing to the secondary decomposition of both ore and country-rock, and to the diffusion of metallic oxides which have stained the whole, it is often

difficult to recognize whether partially replaced portions of the vein-material were originally sandstone or porphyry. The outcrops of no less than four sheets of the latter rock have been traced on the surface; but, owing to the complicated movements along the fissured zone, there would appear at first glance to be many more there; for, not only do the walls change from sandstone to porphyry and *vice versa* in alternate levels, but on one and the same level one finds these rocks alternating with each other in an apparently unaccountable manner in either wall. One can conceive of such a thing as resulting from the movements described above, combined with the gentle inclination of the beds; but to represent it accurately on paper would require most full and accurate surveys and detailed studies along every drift, map in hand.

A second and apparently quite recent dynamic movement is evidenced by a faulting of the vein-fissures in the upper part of the mine. The displacement is slight in amount, and, what is rather unusual, it is along a nearly horizontal plane. As this plane dips gently east, it is probable that it is a movement along a bed of decomposed shale, rendered slippery by the percolating of surface waters.

Similar horizontal faults along stratification-planes were observed by me in some of the gold-veins of the north slopes of Mt. Guyot, east of Breckenridge, Colorado. By these the veins are faulted in a series of gently inclined steps, only a few feet wide, the movement following the stratification-planes. The veins themselves are narrow cracks in a dense blue-black argillaceous shale of Cretaceous age. Owing to the insoluble character of the country-rock, there has been comparatively little replacement of it by the mineral-bearing solutions. Sulphurets of iron and gold have been deposited in narrow fissures, seldom over an inch thick, and by secondary decomposition the iron is oxidized and the gold left in the form of leaf- or wire-gold. Sometimes a mat of interwoven wires of gold, an inch thick and larger than the palm of the hand, is found. Small quantities of gold have been deposited along the minute natural joints of the shales also, so that extraordinarily rich placers have resulted from their disintegration and abrasion by atmospheric agents.

Many more instances of different structural conditions might be presented did space and time allow; but I think the above are sufficient to illustrate the main proposition which I wish to lay down, namely: that careful structural study of the district in which a mine

occurs and of the manner in which the water-passages were formed, which originally gave access to the mineral-bearing solutions, is of the greatest importance to the mining engineer in his determination of the probable extent and value of a deposit and of the best method of exploiting it. Hence, that he must be something of a structural geologist, as well as a technical mining engineer.

NOTES ON THE TOPOGRAPHY AND GEOLOGY OF WESTERN NORTH CAROLINA—THE HIAWASSEE VALLEY.

BY HENRY E. COLTON, MURPHY, CHEROKEE COUNTY, N. C.

(Boston Meeting, February, 1888.)

NEAR the town of Christiansburg, Va., occurs a singular feature in topographical as well as geological structure, which may be said to have an important bearing on a large area to the southwest. The great water-shed ridge, which divides the waters flowing into the St. Lawrence from those flowing into the Atlantic ocean, coming southwestward, forms all of what was the boundary of Maine as agreed upon by the treaty of 1783, and a part of the present boundary, all of the northern New Hampshire boundary, and entering Vermont, continues south of Lake George. Passing irregularly through New York, it enters Pennsylvania, and there bifurcating, sends one prong to the west as a divide between the waters of the lakes and the Ohio, while the other becomes the steady, regular ridge which gave it the name of Alleghany, and thus passes on southwestward through Virginia. At no point has it ever been the geological divide. Its course, commencing in the Archean rocks, has passed through nearly every series of formation, and comes into Virginia in the Carboniferous strata. The geological divide—the line between the fossiliferous and the non-fossiliferous rocks—lies, in Canada, to the west of it, but from New York south into Virginia is far to the east. Reference to Dr. Hitchcock's map will show this. But at the locality noted these positions are changed; the water-divide becomes the eastern chain, and the geological divide is the western, returning, it may be said, to the positions occupied in the region where they originated. These relations are shown on the large map accompanying the present paper.

For several miles, the track of the Norfolk and Western Railroad

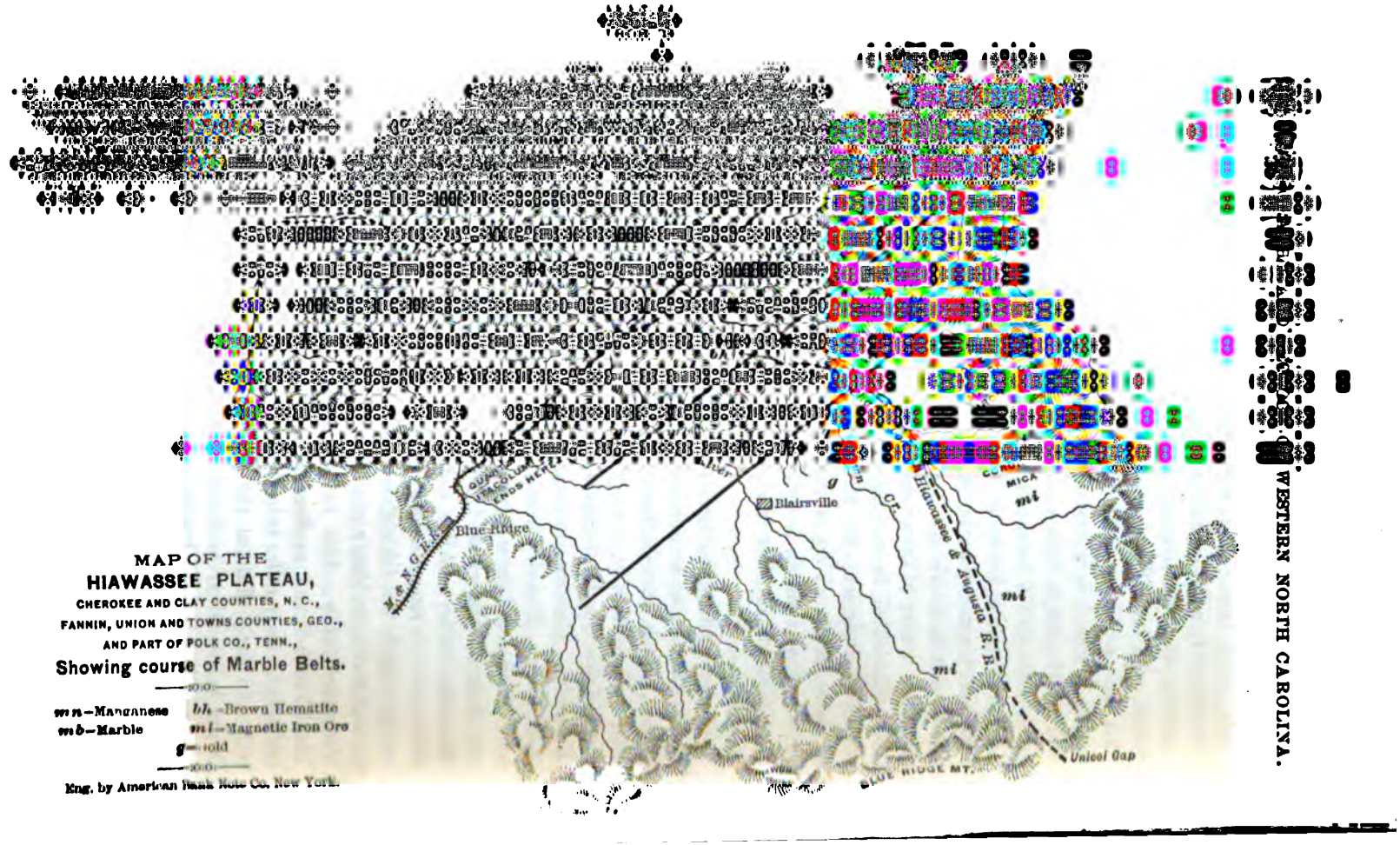
is immediately on the water-dividing ridge, and the interesting fact may be seen, that the waters of the drain on the north of the trackway go into the Roanoke and thus to the Atlantic, while those on the south go into a tributary of New river, and thus to the Ohio. From that point the well-defined mountain ranges continue in a southwest course, the western being ever the divide between the Archean rocks, Huronian and Laurentian, and the lower Silurian formation, but cut through by numerous streams; the eastern never in any sense a geological divide, but at no point, from its commencement east of Christiansburg to its virtual end near Macon, Ga., ever cut across by any stream of water. Called the Blue Ridge in Virginia, where it is the geological but never a water-divide, the name is retained even to its faintest perceptibility as an elevation. Frequently surrounded by mountains vastly higher, and apparently lost in some of the wonderful plateaus peculiar to this region, it can ever be traced as the "endless ridge," the "parting of the waters."

In the city of Atlanta, through the center of which it passes, are houses which shed the rain from their front roofs into the Chattahoochee and the Gulf, while from the rear roofs the water goes into the Ocmulgee and the ocean. The Macon and Western Railroad is built on this ridge, and does not cross a single stream of water from Atlanta to Macon.

Near the boundary line of North Carolina, in Rabun county, Ga. a ridge branches off from the great water-divide, which, running irregularly in a westerly course, crossing the Azoic formation and the geological-divide range, passes into the Silurian limestone a little south of Cleveland, Tenn., and continues southwestward, forming the divide between the waters flowing into the Gulf of Mexico and those flowing into the Mississippi. Thus the territory between the sharp angle made at Christiansburg and this divide forms a long-sided triangle or half ellipse, with irregular sides. In it, between these two great and continuous ranges, is a region, the mineralogy of which is less known than that of any other part of the United States. It is an area varying from twenty miles wide on the Virginia line to seventy miles on the line between North Carolina and Georgia. In it are numerous mountain peaks higher than any others east of the Mississippi, and valleys of wonderful beauty and fertility. It contains great bodies of magnetic ore and of copper-ore, which are continuous from Floyd county, Va., on the western side, to Fannin county, Ga., and on the eastern side to Rabun and Towns counties, Ga. In it have been found and are still worked vast deposits of

mica, which have yielded enormous profits to good management. In it is the only great corundum mine on the American continent, and nowhere else have so many or such a variety of precious stones been found. In its valley and on its hillsides are numerous evidences that these treasures are not now for the first time worked by human hands. Proof is abundant that years ago the mound-builders worked the magic "glass-rock" far into the earth; and, later, that some more skilled hand sought there for the precious metals. The Indian of to-day knows not of either; and whether Hernando De Soto and his men dug the shafts in the valleys of the Hiawassee and the Konetah, which are a source of mysterious wonder, will ever remain a matter of theory and conjecture. They were dug by persons well skilled in mining, and the localities were selected with excellent judgment for the occurrence of veins of ores of the precious metals. Five of those which have come under my observation were sunk on a vein of black oxide of manganese, which the Spaniards, or other workers, undoubtedly thought was sulphuret of silver.

The long triangular territory is divided into seven distinct areas by cross-ranges of mountains, and each of these has its distinctive stream, which, with its lateral branches, forms a distinct water-shed, and, topographically, a sort of table-land. The northern area is drained by the head-waters of New river, which flows into the Kanawha. Coming southward the great streams are the Wautauga, Nolachucky, French Broad, Pigeon, Little Tennessee, Hiawassee and Ocoee. All these streams flow to the north or northwest, and cut through the great Unacoi chain of mountains into the East Tennessee valley. All of them, except the Hiawassee, have towards their source a long distance of placid water in a table-land valley, comparatively but little above their water-level, and thus flowing for some distance, suddenly plunge down to the approximate level of the Tennessee valley. One instance of this is all that is necessary; The French Broad, at Brenard, near its head, has an elevation of 2109 feet; forty miles below, at Asheville, it is 1964 feet, a fall of only 145 feet; while at the Warm Springs, forty-two miles farther down, it has fallen to 1325 feet, being 639 feet fall. There are many miles above Asheville where the turbulent stream, which the traveller on the railroad sees, has scarcely any current, and a little steamboat has for years navigated its waters. The Hiawassee, on the contrary, has no sudden falls, but descends gradually from its source, at Indian Grove Gap, in the cross-divide, to the mouth of Conasauga


MAP OF THE
HIAWASSEE PLATEAU,
CHEROKEE AND CLAY COUNTIES, N. C.,
FANNIN, UNION AND TOWNS COUNTIES, GEO.,
AND PART OF POLK CO., TENN.,
Showing course of Marble Belts.
mn—Manganese
mb—Marble
bh—Brown Hematite
mi—Magnetic Iron Ore
g—Gold
Engd. by American Bank Note Co. New York.
Blue Ridge
Blairsville
Unicoi Gap
BLUE RIDGE MT.
MICA
WESTERN NORTH CAROLINA.

Creek, in the East Tennessee Valley. Its fall in 100 miles is only 1268 feet. In geological features its valley is also different from the areas drained by the other streams named. It is of the region it drains and of this fact that I propose to treat specially. I have, in introducing my subject, spoken thus elaborately of this unknown country, hoping that some of the scientists whose eyes this may reach will give it the careful study it deserves, and which is not permitted by the pressure of my own practical and economic work.

In the small map, Fig. 1, I have shown the area of which I specially propose to treat. It is comprised in the counties of Cherokee and Clay and part of Graham, N. C., and Towns and Fannin, Ga. Of this area I have made two sections: one parallel with the Georgia State line, commencing west of Sequatchie valley and passing just north of Chattanooga, the other farther north, running across Valley River valley about fifteen miles N. E. of Murphy, and for comparison a section across the East Tennessee valley and through the Warm Springs. The whole series of rocks, from the Carboniferous to the oldest Laurentian, can be seen from these sections. It is further seen that at no point between the geological and water-divides is there any limestone found except in the Hiawassee section, and that there it occurs in a series of lines alternating with other rock-strata and metal ores in a regular course of succession. A geological map of North Carolina, now before me, classes the region northwest of the Walnut mountain in Madison county and thence on southwest through Cherokee to the Georgia line as Huronian; while the fact is that from Little Bald to Big Bluff, northwest of the Walnut mountain, including the Warm Springs, the whole area is Lower Silurian. The limestone of the Warm Springs is a cherty Quebec dolomite, and the writer has specimens of it blasted from the bottom of the pool when it was being deepened and enlarged in 1886. A geologist of some note has written that the Warm Springs limestone is the same as the marble of Cherokee. It has no sort of resemblance to that marble, and no more belongs to the same formation than does the granite of Quincy.

The section, Fig. 2, should show the strata more highly inclined, the dip being 80 to 85 degrees to the southeast. In these steeply pitched strata occur veins of manganese-ore, limonite or brown hematite, the marble above alluded to, talc, magnetic and specular iron-ore, chrome-ore, nickel-ore, mica, corundum and other minerals. The dip of the strata and of the veins is to the southeast and the course approximately southwest.

Section up Hiawassee river, from two miles below Murphy, N. C, to Towns Co., Ga. (Valley river is shown on the east of the first quartzite ridge. It cuts through this ridge, changing its course from N.-S. to E.-W., and crossing the first marble belt, joins the Hiawassee. See Fig. 1.)

I have called these ore-bodies veins: some might designate them as beds. Nowhere else have I ever seen manganese-ore and limonite, or hydrated oxide of iron, in anything that resembled a well-defined continuous vein, even one lying between (not across) the strata and conformable to them, and I am aware that many of my readers will doubt the fact; yet fact it is, as can be proved by the testimony of some of our first scientific men and most practical iron-workers, derived from their own personal observation. I admit that, so far as I have been able to ascertain, these ores do not exist in this position beyond the peculiar walled-in area I have mapped, either to the northeast or southwest. And it is further a fact that the crystallization of the marble changes south of the ridge and water-divide on the latter side. I do not attempt to explain these peculiar freaks in this ore-area, but their existence can be proved at any time to one who will visit Cherokee county, N. C.

On the map, quartzite and quartzitic itacolumite are named. These rocks occur in a steep wall-like position, capping the summit of persistent ridges; but their development is greatest, and the ridges containing them are highest, on the Hiawassee river, about midway between the northeast and southwest boundary of the area noted; and there, too, are the best developments of ores. In both directions, northeast and southwest, the height of the ridges containing these quartzites becomes lower and these rocks less prominent; at some localities, shown only by loose rocks on the surface. These quartzites bound the series of strata of ores and marbles, and as the rocks disappear so does the vein-character and quantity of the minerals. What intimate relations they bear to each other, must be the problem for some student other than myself. I simply state the facts as there indicated, that the quartzite walls do not go beyond the encircling ridges; that where they are highest and boldest the ores and marbles are most abundant; that the ores do not go beyond the encircling ridges as defined veins, as in Cherokee; and that the marbles going beyond have a much coarser crystallization, and are nowhere so purely white.

With one exception, the ores approaching the "Bessemer limit" have not been opened to any extent. The exception is a singular gray magnetic ore at Valleytown. This vein of ore has been fairly well developed and traced. At the opening near Valleytown it is solid, 18 feet thick, and dips 85 degrees to the southeast. The same ore outcrops at intervals for many miles; and there is every prospect of its being continuous in length and depth. The wall on one side

is a fine-grained quartzite; the other has not been reached so as to be fully determined, but so far as gone down on is a manganiferous limonite. Two analyses of this ore, the average of many tons on the dump, give:

Iron,	65.56	64.89
Phosphorus,	0.095	0.092
Silica,	not determined	1.29

The manganese-ores occur in two formations, or positions in the strata, and are as distinct in appearance as the rocks in which they are found. One, a highly crystalline variety, is never found except near to or immediately joining the marble-beds; the other never appears where the marble does, but has for its wall-rock on one side a yellowish talco-micaceous slate, while the other side is near to a quartzite ledge. This last has a slight blue tint, and when freshly broken resembles fine-grained tool-steel. I have not traced the continuous line of these ores as perfectly as the outcrops just mentioned, of the brown hematites, but their frequent occurrence in the same position in direct lines gives the impression that they are equally continuous with the other ores. A series of analyses of these ores from various localities is as follows:

	Manganese.	Phosphorus.
1,	43.58	0.056
2,	43.16	0.057
3,	49.39	0.311
4,	44.108	0.141
5,	55.929	0.078
6,	50.770	0.077

Here are ores excellently adapted for the manufacture of spiegeleisen or ferro-manganese, and so far as observation, and, I admit, limited explorations indicate, the quantity is very large.

The magnetic ores, other than that mentioned above, and the hematites and specular iron-ores have not been explored, but the outcrops indicate large and permanent veins of good quality. These ores are not limited to the encircling wall noted. They can be traced through all water-divides and barriers from Virginia to Cranberry, and thence southwestward. They follow alongside of the great chrysolite dike, which also carries the nickel- and chrome-ores. Want of transportation has prevented the development of these ores. To all appearances there are millions of tons of them within 125 miles of Chattanooga.

In the same region where the magnetites and hematites are most abundant (the country around the headwaters of the Hiawassee) are also found copper-ore, chrome-ore, corundum, nickel, mica and asbestos. Two samples of chromate of iron give:

	Iron.	Chromium.
1,	23.43	30.10
2,	30.10	20.34

A highly magnetic specimen of titanic iron-ore, from a vein six feet thick gave:*

Iron,	65.9
Titanium,	4.8
Phosphorus,	none

A vein of slaty hematite, opened to a depth of 5 feet, and showing a width of 10 feet, gave:

Iron,	36.010
Phosphorus,	0.069

As this vein had a very slight show on the surface, it may possibly increase in width and quality with further depth. It is favorably located for mining. It resembles some of the Lake Superior ores.

As so far developed, the great wealth of this region is in its brown hematite iron-ores and its marble. It is the opinion of many experienced iron men of large observation that there is not an equal area in the world where there exists so much brown hematite iron-ore. The amount of it is simply enormous. The fact cannot be expressed in any other words. There are at least five parallel veins of it, and I have had these cut across at intervals along a distance of 30 to 35 miles. Number one of these veins is at no point less than 15 feet thick, and at one place, where opened, there is 35 feet of solid ore by actual measurement with a tape. At another place not opened, it has the appearance of being much thicker. Number two ranges from 12 to 25 feet thick. Number three is from 16 to 18 feet, and number four from 10 feet upwards, and is a very rich ore, slightly magnetic. Number five is from 16 to 28 feet thick, and may have at some localities a greater thickness. I do not claim that these ores are continuously of one thickness. It is admitted that they are lenticular, but so far as careful exploration proves they are never less than 10 feet thick.

* The analyses given in this article are by Prof. McCreath, Dr. James A. Burns, of Atlanta, and J. Blodgett Britton.

It is plain to any eye that these veins are continuous on the surface. The question is, How far do they go down? First, it must be remembered that they are bedded in strata dipping at an angle of 85°, and all theory indicates that they will be found to a great depth; secondly, in the wild excitement about copper, many thought that where brown hematite-ore appeared on the surface, it was the outcrop of a copper vein, as that ore was so capped at Ducktown. Hence shafts were sunk on many of these veins in Cherokee, wherein copper was not found, but the fact was made manifest that the iron-ore existed deep down in the earth and increased in thickness and improved in quality. There are still open large shafts, of a depth of 40 feet and over, by which this can be proved.

The position of these ores makes it possible to mine them at very low cost; at many points it would be simply quarrying iron-ore. Murphy, the center of this ore-region, is 94 miles distant from Chattanooga by a survey recently made for a railroad. These ores could be mined at a cost of 30 cents per ton; and if the railroad proposed is built, the freight will not be over 94 cents. Hence, they can be delivered in Chattanooga at $1.24 cents per ton. From an analysis of 23 samples, taken at random by Mr. John Birkinbine, and by no means picked, the result was:

Metallic iron,	50.020
Phosphorus,	.712
Silica,	9.080

Therefore, as Dr. Britton says, two tons should make a ton of iron. By omitting only one of the samples the average of the rest in phosphorus is reduced to 0.3745, while the iron per cent. is increased. Dr. H. S. Fleming presented at the Scranton meeting an excellent paper on "Iron Ores used in the Chattanooga District" (*Trans.*, xv. 757), which is especially valuable from the fact that it was not compiled from the work of varied chemists, but is the result of his own practical experience at the Rockwood, Dayton and other furnaces of that region. He gives the average of certain ores as follows:

	Iron.	Silica.	Phosphorus.
Tennessee river soft ores,	42 80	14.21	0.736
Tennessee river hard ores,	31.47	6.64	0.524
Georgia brown hematite,	44.85	18.56	0.857
Roane Iron Co.'s stock-pile,	48.36	14.78	0.63
Cranberry magnetite, as bought and used by Roane Iron Co.,	47.66	20.53	0.0104

Therefore, the Murphy region brown hematites average a larger percentage of metallic iron, less phosphorus, than any ore but Cranberry, and less silica than any ore used in or around Chattanooga. Mr. Birkinbine will testify that his samples were not in any manner choice: in fact, the owners of the properties were provoked with him for taking so many low-grade specimens.

The value of these red-fossil and brown ores in Chattanooga is now $1.50 to $2.00 per ton. It is plain that the Hiawassee ores can be delivered there at a very fair profit, or at so much lower price as to give Chattanooga furnace-men a great advantage in the manufacture of both common and steel-making pig-iron. It is proper to state that the one ore taken from Mr. Birkinbine's lot of samples, whereby the percentage of phosphorus is so much reduced, is an ore fit probably for no other purpose than the manufacture of basic steel. And a singular fact is that within 100 yards—parallel to it—is a vein of ore low in phosphorus. Some ores from special veins, but all included in the above average, give by analysis:

	No. I.	No. II.	No. III.	No. IV.
Iron, . . .	54.625	52 06	54.17	56.13
Phosphorus, .	0.302	0.14	0 79	0.86

No one of the veins from which the above analyses were made is less than 12 feet thick, and No. I. has been opened in many places, nowhere less than 20 feet wide.

Marble.

The marble of this intra-montane region is generally white. All is of fine-grained crystallization; some, so specially superior in whiteness and texture as to be of the best statuary grade, has been pronounced by experienced experts to be equal to the very finest Italian. The other colors are blue-black and gray-black, mottled and flesh-color. It exists in layers from 3 to 10 feet in thickness, and can be easily quarried. The most searching investigation of many scientific men has never yet found a fossil in it.

Talc.

In close proximity to the marble is found a mineral called "talc" by the people of that region, and so classed in the *Mineral Resources of the United States*. It is a form of silicate of magnesia, pearly white, with a satin luster, and unctuous to the touch. When first quarried, it is easily cut into any form, but becomes very hard by

long exposure to air in a dry place. Two mines are now worked in it; and the demand seems to be greater than the supply.

Development.

The question is in the mind of every one: If all this mineral wealth, why has it not been known and used? First, some of it has been known to a few scientists only; secondly, it is surrounded by a wild mountain region, and until lately the nearest railroad has been over 65 miles distant, by wagon-roads over rough, high and steep mountain ranges; thirdly, it has, from its inaccessibility, been neglected by the North Carolina geologists. Two railroads are now reaching towards Murphy. One, a narrow-gauge, comes from Marietta, Ga., connecting there with the Western & Atlanta. The distance by it to Atlanta is 132 miles; to Chattanooga, 232 miles. The other is called the Murphy Division of the Western North Carolina R.R., and comes from Asheville, connecting there with the W. N. C. R.R., both to the north and east. The distance to Asheville by it is 125 miles. It is now completed and running within 25 miles of Murphy, and work is being done on the remainder. It has been rapidly constructed and with heavy grades, but if improved could be made a valuable road. Neither of these railroads connects with a region which is a market for the mineral wealth of the Hiawasse country. The road needed is one direct to Chattanooga. Charters have been obtained and a line carefully surveyed for such a railroad; and there is great probability that it will be built in a short time. Should this be accomplished, there is certain to be a great future for the Hiawassee Valley. The survey shows that a railroad of standard gauge can be built at a moderate cost, with maximum grades under 65 feet per mile and only a few 6° curves, while there is one tangent 28,000 feet long, and that the road could reach Chattanooga in a distance of 94 miles or less from the iron-ore veins at Murphy. The charters authorize this road to be extended eastward, as it will be, through Clay county, N. C., Towns and Rabun, Ga., to Toccoa, on the Atlanta & Charlotte R.R. It will thus reach immense bodies of Bessemer ores; and Chattanooga will thereby be nearer to Bessemer ores than any coke iron-making city in the United States—fully 135 miles nearer than now from the Cranberry mines. Moreover, the large quantity and excellent quality of the manganese ores make the manufacture of spiegeleisen or ferro-manganese possible at very low cost, either in Cherokee or Chattanooga. Suppose we take the gray magnetite near Valleytown and manganese-ore No. 1, which are

MAP OF

th Carolina and Northeast Georgia,

SHOWING POSITIONS OF

EOLOGICAL AND WATER DIVIDES

AND PECULIAR TOPOGRAPHY OF THE

NORTH CAROLINA MOUNTAIN REGION.

By HENRY E. COLTON, Geologist,

CHATTANOOGA, Tenn.

MILES.

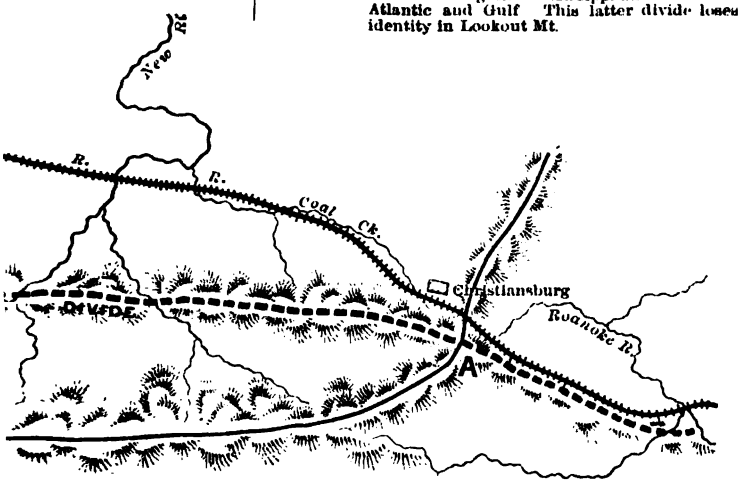
EXPLANATION.

Water Divide.

Geological Divide.

A—Crossing of Geological and Water Divides. Geological Divide comes to the East.

B—2d Crossing of Geological and Divide between waters flowing into Mississippi and those into Atlantic and Gulf. This latter divide loses identity in Lookout Mt.



found close together. One ton of each would make a mixture containing 21.79 manganese, 32.78 iron and 0.075 phosphorus; and assuming two tons of mixture per ton of product, and the same proportion in the product, the ferro-manganese should contain about 40 manganese and 0.151 phosphorus.

Such ferro-manganese as this would be worth at least $40 per ton. A study of the analyses given will show other excellent mixtures. Manganese No. 4 is near the red hematite containing 36.01 of iron and only 0.069 of phosphorus. Wood for charcoal is abundant and to be obtained at low prices, and there are no better sites for furnaces to be found anywhere than near the junction of the Notla and Hiawassee rivers, or the Hiawassee and Valley rivers. At the latter point a furnace could be located where it would be within 50 yards of abundance of ore and limestone and unlimited supply of water, and also within 100 yards of where three railroads will be constructed at no very distant day.

I have herein spoken of quartzitic itacolumite. This may seem a singular term, but I use it for the reason that this apparently flint-like quartzite, in a few miles, frequently changes to a soft sandstone, which, when in layers, is flexible, really a true itacolumite. I have not chosen to discuss the age of these peculiar strata or their relationship with any others. The quartzite sandstone and the marble are both absolutely free from fossils, and there is not the slightest ground upon which to base a relationship between them and the Potsdam sandstones and Knox dolomites of East Tennessee.

THE ELECTRIC MOTOR IN MINING OPERATIONS.

BY GEORGE W. MANSFIELD, MELROSE HIGHLANDS, MASS.

(Boston Meeting, February, 1888.)

MY plan in this paper is, first, to prove three general points, and then to take up the specific applications of the electric motor to mining work. The three points are:

1. The electric system is the simplest method known for transmitting and distributing any amount of energy from one locality to another, irrespective of the intervening distance.

2. The electric system with its applications is not in its infancy.

3. With the electric system, small and large amounts of power can be transmitted economically and profitably short or great distances.

As a simple explanation of the production of electricity in large powers, and the way it is utilized in motors, I will briefly describe a simple form of dynamo and motor. Take a loop of wire, bend it into the shape of a rectangle, support it by its longest axis between the poles of an ordinary permanent horse-shoe magnet, with this axis at right angles to the general length of the magnet, and then rotate it. As the wire cuts through the magnetism emanating from the magnet's poles, electricity will be developed in the wire. If now you multiply these rectangles, winding them lengthwise upon an iron body cylindrical in shape (the wires crossing at the ends, of course), arrange each rectangle so that its ends will come out off from the iron cylinder, connect them so that they will be as one wire, and solder the connected ends to a collector, whereby the electricity can be conducted away from the rotating part, you have a modern first-class armature. This now rotate between the poles of a powerful electro-magnet instead of a permanent steel magnet, and, if you have planned large enough, you will obtain electricity equivalent to 1, 100, or 1000 h. p. What could be more simple! All dynamos are essentially the same, consisting of an armature or rotating part, and the field-magnets or stationary part. Practically, nothing to wear out, or give out. A well-proportioned machine has an efficiency of over 90 per cent., and would outlast, twice over, the steam-engine driving it (if it be driven by this means). To produce electricity by dynamos, then, some ample driving power must be had. Steam, water, wind, and gas have all been used; but the first two only will be considered here.

An electric motor is a dynamo, in every sense of the word; but instead of being driven it uses the electricity and drives; one produces, the other consumes.

Consider now the simplicity of an entire system. A dynamo, producing electricity and driven by steam or water; and two copper wires necessary for the complete metallic circuit, leading from the dynamo across the country, no matter what its nature, to the locality where the motors are located. This is all. Ordinarily, for convenience of application, motors are of a different shape from dynamos, but they could be identical.

The line-wire once put up, in a substantial manner, at a construc-

tional cost of a few hundred dollars per mile, is up for all time, and requires practically no attention.

Compare such a line with piping or wire rope; compare dynamos and motors, with no more machinery about them than an emery-wheel, with your steam or gas-engine, your hydraulic or pneumatic motors—if there are any such—and what can the decision be but favorable to electricity?

I have been speaking of transmission; there are similar advantages in application. There is no such varied capacity in either steam, gas, air, water or cable. Can we hoist, haul, transport, pump, ventilate, drill, blast, light, or heat, by any one of these? We can by electricity, and it is being done every day in the year. It is in this grand capability of subdivision, application, and distribution, for all purposes, that electricity reigns supreme.

Let us now see whether we have passed beyond the experimental stage or not.

Can you produce electricity enough to do heavy work? is a question often asked. In reply, I would say, that there are a score of electric-light stations that are, every night, producing electricity equivalent to 1000 h. p., and over, and distributing it all the way from 50 feet to 50 miles. Here, in Boston, there are about 3000 arc-lights, requiring fully 2500 h. p. of steam-engines. This enormous power is passing every night, in the form of electricity, through the wires which are stretched above the housetops all over the city.

There is no limit to its production or transmission. It is *possible* to transmit the total available power of Niagara Falls, some 7,000,000 h. p., to Boston. But, would it pay? is the question to solve. Taking all the electric-lights throughout the country, and roughly figuring, we find that there is fully 250,000 h. p. engaged in producing electricity every night in the year. If there were an assured return for money advanced, this vast amount could be condensed into one station. According to the Tenth Census, thirteen of our western States and Territories, those producing most of our ores, excluding Pennsylvania, use only 211,840 h. p. in all their productions, mining or otherwise. Speaking of mining only, I think it safe to say, that for every h. p. used in mining there are ten employed in producing electricity. Financially, there is fully $200,000,000 invested in the manufacture and use of electric apparatus for the mechanical production of electric currents for lighting and power. I bring forth these figures to illustrate that the use of electricity is not in its infancy.

A few words, now, as to what has been accomplished in transmitting power various distances. An arc-light consumes about .8 of a horse-power of electric energy; now there are almost innumerable circuits from 10 to 50 miles long throughout our country with arc-lights indiscriminately placed their *entire* length. This is practically transmitting power; for a motor could be placed where any one of the lights is. Abroad, Marcel Deprez has transmitted 40 h. p. through 70 miles of wire, with a commercial efficiency of 56 per cent. Mr. Brown, in Switzerland, has put into practical operation a plant transmitting 50 h. p. 5 miles, with a commercial efficiency of over 70 per cent. The Thomson-Houston dynamos are over 90 per cent. efficiency, as are also their motors. Therefore, if a steam-engine dynamo and motor were placed side by side we would obtain, of the 100 indicated h. p. of the engine, over 81 h. p. in mechanical power from the motor.

To whatever distance we remove the motor from the dynamo we lose power in the circuit. Copper is the best conductor of electricity, and yet this offers resistance. To decrease this resistance, we have to increase the conductors in size. This point is similar to the general practical rules governing hydraulic transmission.

To illustrate in a comprehensive manner the cost of transmission, I have worked out several cases, and have tabulated them with numerous additional figures of interest:

Tables showing Cost of Complete Electric Plant for Transmission of Power, Various Distances. Potential at Central Station, 560 *Volts. Loss on Line,* 10 *Per Cent. Copper Wire, going and return.*

Horse-Power Transmitted.	1 Mile.				5 Miles.			
	Cost of Wire.	Cost of Line Construction.	Cost of Dynamos and Motors.	Total Cost of Plant.	Cost of Wire.	Cost of Line Construction.	Cost of Dynamos and Motors.	Total Cost Electric Plant.
1	25	175	250	450	500	1,000	250	1,750
5	100	175	875	1,150	2,500	1,000	875	4,375
10	200	200	1,750	2,150	5,000	1,100	1,750	7,850
25	500	200	4,350	5,050	12,500	1,100	4,350	17,950
50	1,000	250	8,750	10,000	25,000	1,300	8,750	35,050
100	2,000	250	17,500	19,750	50,000	1,300	17,500	68,800
500	10,000	300	62,500	72,800	250,000	1,600	62,500	314,100

Horse-Power Transmitted.	10 Miles.				25 Miles.			
	Cost of Wire.	Cost of Line.	Cost of Dynamos and Motors.	Total Cost.	Cost of Wire.	Cost of Line.	Cost of Dynamos and Motors.	Total Cost.
1	2,000	2,000	250	4,250	12,500	7,500	250	20,250
5	10,000	2,000	875	12,875	62,500	7,500	875	70,875
10	20,000	2,200	1,750	24,000	125,000	10,000	1,750	136,750
25	50,000	2,500	4,350	56,850	312,500	10,000	4,350	326,850
50	100,000	3,000	8,750	111,750	625,000	12,500	8,750	646,250
100	200,000	3,500	17,500	221,000	1,250,000	13,750	17,500	1,281,250
500	1,000,000	4,000	62,500	1,066,500	6,250,000	15,000	62,500	6,327,500

Horse-Power Transmitted.	50 Miles.				100 Miles.			
	Cost of Wire.	Cost of Line Construction.	Cost of Dynamos and Motors.	Total Cost of Plant.	Cost of Wire.	Cost of Line.	Cost of Dynamos and Motors.	Total Cost of Electric Plant.
1	50,000	15,000	250	65.250	200,000	30,000	350	230,250
5	250,000	15,000	875	265,875	1,000,000	30,000	875	1,030,875
10	500,000	20,000	1,750	521,750	2,000,000	40,000	1,750	2,041,750
25	1,250,000	20,000	4,350	1,274,350	5,000,000	40,000	4,350	5,044,350
50	2,500,000	25,000	8,750	2,533,750	10,000,000	50,000	8,750	10,058,750
100	5,000,000	27,500	17,500	5,045,000	20,000,000	55,000	17,500	20.072,500
500	25,000,000	30,000	62,500	25,092,500	40,000,000	60,000	62,500	40,122,500

I have calculated these figures on the arbitrary potential of 560 volts, which is very low for the transmission of large powers; nevertheless, it was necessary to assume some basis. This potential is generally used to-day for power-circuits.

It will be noticed that the cost of copper is the principal item. To reduce this, I would state the following electrical fact:

The cost of copper decreases as the square of the potential increases. Take, for instance, the extreme case in the last table, viz., 500 h. p. transmitted 100 miles. If we double our potential, making it 1000 volts, the cost of our copper will decrease to one-quarter, or $10,000,000. If, now, we again double, we decrease to one-quarter again, or, at 2000 volts potential the cost of copper would be $2,500,000. The total cost of the electric plant at this potential is $2,622,500. Obviously, there is a limit to the increase of potential; and in the transmission of such enormous powers, I would place that limit at one more doubling, or 4000 volts. This is a possible voltage, for there are many electric-light circuits in our large cities to-day of this potential, and even higher.

1000 volts. is a safe and easily handled potential; and if this

potential were used, a large saving would evidently be effected. Professor Thomson has devised and patented a method whereby very high potentials can be used to overcome distance, and at the receiving station be reduced to lower safe-working potentials. This plainly means a tremendous saving in cost of copper.

Having briefly proved the three essential points started with, I will now take up, more specifically, the application of the electric motor to the various wants of the mining engineer.

For simplicity, I have divided the subject-matter into various headings, as follows:

1. Drilling and excavating.
2. Hoisting.
3. Hauling.
4. Transporting.
5. Pumping.
6. Ventilating.
7. All other mechanical work (stamps, crushers, amalgamators, etc.)

1. Drilling and Excavating.

Motors have already been used most successfully under this head, both drilling by impact and boring with diamond drills. The latter method was successfully used several years ago by M. Taverdon, in France, and in his experiments he was fully able to compete with steam or compressed air. Patents are being granted, and it is absolutely certain that this method will have an enormous development. In rapid tunneling, in running adit-levels, in deep and crooked workings, or in all work requiring hasty construction, the great trouble and expense of shifting and relaying pipes, with great losses due to leakage, etc., is avoided. The necessary wires simply cleated to wall or posts as work advances, thus always being entirely out of the way. In surface-work, the area over which we can work before changing steam-plant is vastly increased. In any direction, over rocks, hills, or gullies, on a straight line run the wires, for miles if desired, and drilling and boring can still go on. In excavating, the cranes, shovels, scrapers, etc., are all run with other motors, taking power from the same wires. These applications have already been made, hence it is needless to enlarge. One other point: the wires once in place, none but casual inspection is needed to keep them in order; whereas, with compressed air, steam, or hydraulic-power, a set of men is required to be constantly on hand, repairing, changing,

etc. Furthermore, the room which pipes, etc., occupy in the shafts is by no means small, and is an important item to be considered as compared with wires.

In competition with electricity under this head there can be mentioned only two systems—compressed air and steam. Considering the former, its excessive first cost puts it out of the comparison, except in underground work Here its redeeming feature is the introduction of fresh air. This is, no doubt, a valuable adjunct; but a mine ventilated by electricity, as it could be if electricity was used for drilling, etc., would not require compressed air for the benefit of the men.

It can be proven that the original cost of a compressed-air plant and its maintenance is much greater than that of the electric system, and that, for large work, it is not as economical to operate.

For short distances only, and in open work, steam may be more advantageously employed; beyond these conditions it cannot be considered a competitor.

2. Hoisting.

With an abundance of water-power, located within a few miles of your mine, there is no reason why we cannot hoist entirely by transmitted electricity as economically as by steam. The variety of mines, and their natural surroundings, are so diversified, that it is difficult to give one or two illustrations to cover so much ground.

I shall divide into two general headings—shallow and deep mines.

In all placer-mining, and mines of but a few hundred feet in depth, where there is uncertainty as to the location of workings, or the permanence of the location temporarily selected; in all preliminary explorations, etc., the electric motor is so easily moved and applied, and its scope so great, that it is vastly cheaper and better than any other means, beyond certain limits. In very deep and large mines, however, permanently located, probably steam, acting direct, is to be preferred.

The case, although not exactly similar, is analogous to the application of motors to elevators in our buildings. There are scores applied to this purpose, hoisting many tons in weight, and with speeds from 30 to 130 feet, and more, a minute; many of these have been in daily operation for from three to four years, always giving the highest satisfaction.

3. Hauling.

Under this head too much cannot be said, but I shall be brief.

The application to street railways bears a strong resemblance to the hauling of ore; therefore, I will mention what has been accomplished in this direction. There are in the aggregate, in this country, about 125 miles of railways so operated to-day; the motors propelling loads of 10 tons, at speeds of 1 to 15 miles per hour, to distances of 8 miles, and over grades of 10 per cent. Probably 2,000,000 passengers are now annually carried on these roads. As to size, let me mention that three years ago, I had charge of the experiments then made on the Elevated Railroad of New York City. The weight of the motor was 9½ tons, and capacity about 100 h. p. With this motor the four regular coaches, nearly full of passengers, were drawn over a distance of two miles, and up grades of 2 per cent., at an average speed of eight miles per hour. I believe this is the largest experiment of its kind that has ever taken place in this country.

In the application to mining work, the principles involved are plainly the same, the change being only in detail. There have been already numerous applications abroad, and one in this country. The best known application to mining haulage was made in 1882, at the Zankerode pit, near Dresden, in Saxony. They employ there some 800 wagons, each of about 3 cubic feet capacity. The motors are of 6 h. p. capacity, and haul trains of from 10 to 20 wagons, maximum weight 13 tons, at a speed of about 5 miles per hour. The length of track is ⅛ mile. The cost of operation has been about 1½ cent per ton. The total cost of the plant was about $4000.

An interesting comparison from actual practice shows the ratio of cost of operating to be as follows:

Electric Traction.	Horse.	Man.
2.7	3.7	6.2

Since the above tests were made, great improvements have been made in the efficiencies of motors; hence, although the above is an excellent showing, a still better one could be made to-day. The efficiencies to-day of good motors range from 85 to 95 per cent.

Dr. Hopkinson, an eminent English scientist, from very careful tests of the Ressbrook and Newry Railroad, found that its electrical efficiency was 72.7 per cent. From other sources we have proof that this figure is by no means excessive; and it is safe to say, that a plant installed with ordinary care in any mine would give from 60 to 70

per cent. electrical efficiency. I place the figures for mining work lower than in surface tramway-work, since it is reasonable to suppose, from conditions commonly existing in mines, that not as excellent results could be obtained as from a surface-road.

The only systems to be compared with the electric under this heading are those of the locomotive and the wire rope. In construction, the locomotive system requires nothing but the rails; electricity, the addition of an overhead wire, easily placed, and requiring practically no attention; and the wire rope, a line of sheave-wheels, upon the floor of the level, upon which rests the rope, the whole squarely in the way of any other traffic in the same level. Furthermore, with wire ropes, the room required at the mouth of level or incline for the gears, winding-drums, engine, etc.; the constant expense of repair and maintenance; the short life of the cable, its liability to breakage and consequent stoppage of the entire line; its inapplicability for crooked workings, and, lastly, its enormous waste of power (since there never is obtained in useful work overcoming resistance of traction and grades, over 25 or 30 per cent. of the prime source of power)—all unite to render its adoption for mining work almost absurd, in view of the simplicity and practicability of the electric system. Except for hoisting, there is practically no place for wire ropes in mining operations.

The steam-engine worries with its complications and constant attendance; its heat; its smoke and gases, vitiating the air; and, lastly, its fire-setting ability. For haulage, the steam-engine in mines is to be avoided. Through the adoption of the electric motor all these worriments disappear.

4. Transporting.

Under this head I wish to simply mention the few recent developments relating to what is known as the telperage system. Wire ropes are stretched from post to post across country as desired; depending from these ropes are cars; the motor is placed in one of these cars, and drives up to the driving-wheels by means of belts, chains, or gears.

Several different plans have been devised, as to position of ropes and cars, but essentially they are all alike.

It is beyond all question that there is a very prolific field for application in this direction, and the stimulus necessary is the only thing now lacking for a general adoption in scores of places.

Abroad, the idea has been put into practical shape by those two eminent electricians, Professors Ayrton and Perry. The line at Glynde, England, is about one mile long, and used to transport clay. Trains of 8 or 10 skips are ordinarily made up, and when loaded the total weight is two tons. In every way it was most practically and thoroughly tested, and in every way it has given perfect satisfaction.

5. Pumping.

This heading is, unquestionably, a most important one; and one which demands a great deal of careful consideration on the part of the mining engineer. For practical working, the only methods now employed to any considerable extent are those of direct-acting steam-pumps and pumps operated from the surface through wooden beams.

Compressed air is too expensive, and there is no practical motor for driving the pumps.

Wire ropes are likewise expensive and inapplicable; and any hydraulic method is away out of the question. Steam, as mentioned above, is employed—but under what difficulties and expense! Consider the piping, the maintenance, and the excessive loss due to radiation. Furthermore, for any considerable distance steam is entirely impractical.

The practical application of the electric-motor is simple, since most manufacturers of pumps adapt them for belts or gears. The only thing, then, to provide for is a belt or a pinion for the armature-shaft. The intervening distance between the power-station, on the surface, and the point where the pump is needed, is of small consideration as regards construction, for copper wires can be easily run irrespective of distance or conditions. A plant, complete, consisting of 30-h. p. dynamo, 20-h. p. motor, with wires, insulators, etc., would cost about $3200, and would be capable of doing—one mile distant from station—20 h. p. actual work in pumping water.

A number of applications have already been made abroad; notably at St. John's colliery, Normantown, and at the Thallern colliery, on the Danube. Another application is a small plant at the Trafalgar colliery, England: here, the duty of the motor was to pump 114 gallons per minute through 3900 feet of 7-inch pipe, with a lift of 300 feet. The motor capacity was 10 h. p.; the operating expenses, per day of 22 hours, was $35; total cost of installation, $3220; efficiency, 45 per cent.—*i. e.*, of the belt-power from engine to water thrown out, only 55 per cent. was lost. The cost of such a plant to-day would be undoubtedly diminished one-half.

The wooden-beam method mentioned, working in connection with a plunger-pump, has no claims to convenience or economy, and in these days is simply used in the capacity of a cheap make-shift. Consider the room it necessitates; consider the limited distance through which it can be operated, and finally consider its efficiency, and no one can think of it in comparison with electricity.

Taking all things into consideration, I would unhesitatingly agree to guarantee, outside of certain limits, two gallons pumped by electricity for every one now pumped by any other existing system for the same cost and expense of maintenance.

6. Ventilating.

Upon this topic much can be said; but no particularly new application of electricity is called for. There are hundreds of large fans in daily operation throughout our country, and there is no reason why they cannot be used in mines; in point of fact, I consider it the only proper method whereby a mine can be ventilated. In large mines, especially in the coal-regions, an enormous suction-fan is placed at the mouth of the up-cast shaft, and the air is sucked through the entire system. Evidently where the men are working at the various headings, and in the stalls or rooms, the only way fresh air is obtained is through the construction of wooden boxes leading air in. This is extremely expensive, not only to construct but to maintain.

It seems to me that by the proper placing of numerous motors, each attached to a fan of the well-known Wing or Blackman form, a constant circulation can be maintained.

By dividing a mine into sections, and then by placing in the partitions between each section electric fans, all propelling the air in the same direction, a perfect system is to be had. Plainly, fresh air must be drawn in somewhere and the foul air expelled at another shaft.

For ventilating chambers, a small motor can be placed at the entrance of each near the top, blowing the air out into the general level, along which the main body of air is propelled. In principle, it may be somewhat similar to the first plan I have just mentioned, but you overcome the tremendous frictional losses.

All previous general remarks as to convenience and simplicity that I have made upon the electric motor will apply here, and I will not repeat them. Certain it is, that there is a great field for study

and application in this direction. Abroad, several mines are already ventilated through the agency of electric motors.

7. All Other Work.

It is unnecessary to say much under this head, for it seems to me that what has been said already sufficiently shows that the adaptability of the electric motor is universal. It is a positive fact that any mine in this country, with all its varied mechanical appliances, could be operated entirely by electric power; and I have no doubt that within a few years such a general application will have been made.

In concluding this general paper, I would indicate, as what seems to me to be the pith of the application of electricity to mining operations, the concentration of the entire production of power in one building; the facilities of distribution, irrespective of distance and difficulties, and the comprehensiveness of the applications of the same form of motive power.

SYSTEMS OF MINING IN LARGE BODIES OF SOFT ORE.

BY RICHARD P. ROTHWELL, NEW YORK CITY.

(Boston Meeting, February, 1888.)

At the last meeting of the Institute, Mr. Per Larsson read a very interesting paper on the Chapin mine of Michigan, in which he described the systems of mining that had, at different times, been tried in that great ore-body, and especially the system now in use, which consists in working the ore out from the bottom upward in lifts. Shortly after that paper was written, but before it had been published, or that I had had an opportunity of reading it, I had the pleasure of visiting the mine. At that time I was making a professional examination of a number of the mines on the Gogebic Range, and others in the Menominee region, Michigan and Wisconsin, and had occasion then, as during several previous visits, to study closely the methods and cost of mining in these districts. As the outcome of these investigations, it appeared evident to me that the system of working on the chamber-and-pillar plan, which up to that time had been the only method of underground work adopted in the Gogebic

mines, must inevitably before long be abandoned, a result already arrived at in the Chapin mine; and I suggested to my clients the adoption of a modification, described in the following paper, of the old method used in some of the large coal-beds in France. My impression at the time was, that this method had not yet been applied in the iron-ore mines; but in answer to some remarks I made in the *Engineering and Mining Journal*, when I published Mr. Larsson's paper, I received from Mr. James H. Goudie, of Ironwood, Mich., a description of a system of mining in use in one of the large iron-ore mines in the North of England, which I published in the *Engineering and Mining Journal*, 24th December, 1887. This method of mining, though quite different in detail, still embodies the same principles as that I had advocated.

It has been brought to my attention since preparing this paper, and, in fact, on my way to this meeting, that at Longdale, Va., essentially the same method of work, namely, working out the ore-body from the top down, has for some years been in very successful use. I am very pleased to cite these cases of successful application of the method I had proposed, though they make the application of the system to ore-mining less novel than I had supposed it to be. Nevertheless, since the details of the method I have described here are, I believe, different from those heretofore proposed in some rather important particulars, and as they may be suggestive to other engineers, I gladly contribute it for what it is worth.

The objections which suggest themselves on reading the description given by Mr. Larsson of the system in use at Chapin mine, and still more forcibly on examining the mine, are:

First, the enormous amount of dead-work, rock-drifts, "mills," cross-cuts, etc., required, and which add so heavily to the cost of the ore. Ninety per cent. of the dead-work is said to be in barren ground.

Second. This method of working where the ore is hard enough to stand without timbering over the width of a room, is much more expensive than that practiced at the Iron River mine (which I shall presently describe), and, where the ground is soft, it presents the very serious difficulty of working on a sinking packing under foot, and with a moving top; for, as the timbering has to stand on the filling, it not only sinks unevenly, and thus breaks much timber, but the ore above it sinks and becomes loose and slippery, requiring very close and heavy timbering and fore-poling, even in making a cross-cut of eight or nine feet wide. The timber is all lost, and it is even diffi-

cult to keep open the little drift on the foot-wall that extends fifty feet from one cross-cut to the other.

In answer to an inquiry as to how much the ore in the vein would come down or follow the sinking of the packing in working up a lift of 100 feet, I was told "about 10 feet." It is easy to imagine what a difficult or impossible thing it is to maintain any kind of regularity under such conditions, or to work with any comfort with a slippery, sinking roof and a settling floor.

System of Mining at the Iron River Mine.

The system of mining in use at the Iron River mine, Menominee region, is extremely well adapted to ores hard enough to stand over the width of the vein. The accompanying section (Fig. 1) across a stope, shows the method of work. The ore is taken down in an overhand stope running the width of the vein for any desirable length, and for a height of say 12 feet. A timber-drift is then built along the floor of the stope, and the balance of the stope is packed with waste sent down from the surface through winzes previously sunk or upraised at intervals of about 50 feet in the length of the stope.

A "mill," or "chute," is carried up every 50 feet along the level to run the ore down, and is made about 4 feet square inside, and is built of round, rough hardwood sticks.

The spaces between the sticks when they do not fit closely are filled with pieces of plank, and the inside of the "mill" is lined with hardwood planks spiked on to the side timbers. These planks are easily replaced when worn.

The packing is levelled over as close to the backs of the stope as is convenient for working, and is planked over, to keep the ore from mixing with the filling.

The "mill" is carried up before the filling as high as this is to go, and when the filling is levelled off a few large sticks are laid across the mill, leaving intervals large enough to throw the ore down, but narrow enough to prevent the falling in of a man or a block of ore large enough to choke up the outlet from the mill.

This method of work is extremely satisfactory where the ore is strong enough to stand without timbering across the vein. It requires no timber except the level and the mills, and the use of some

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at 13 cents per ton

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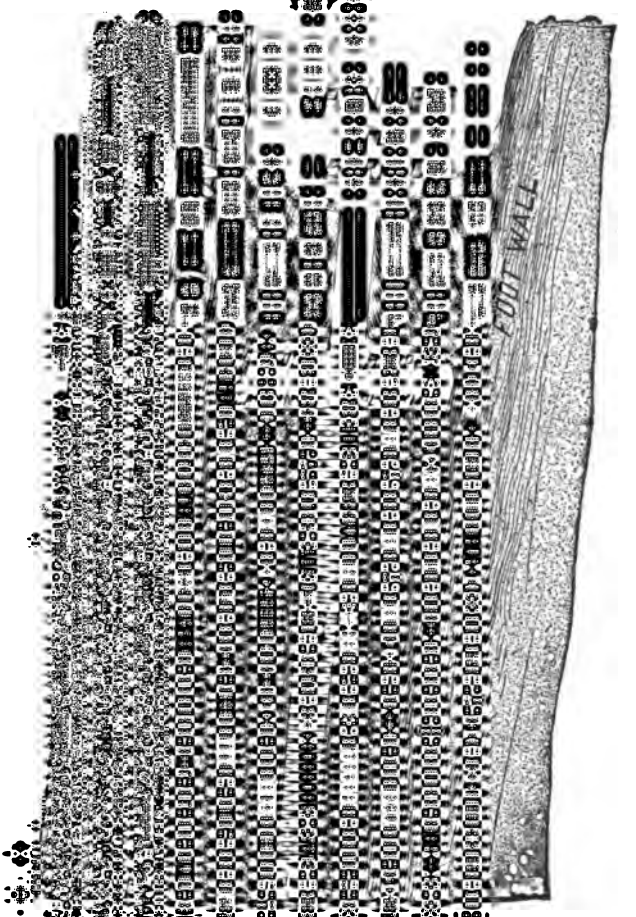

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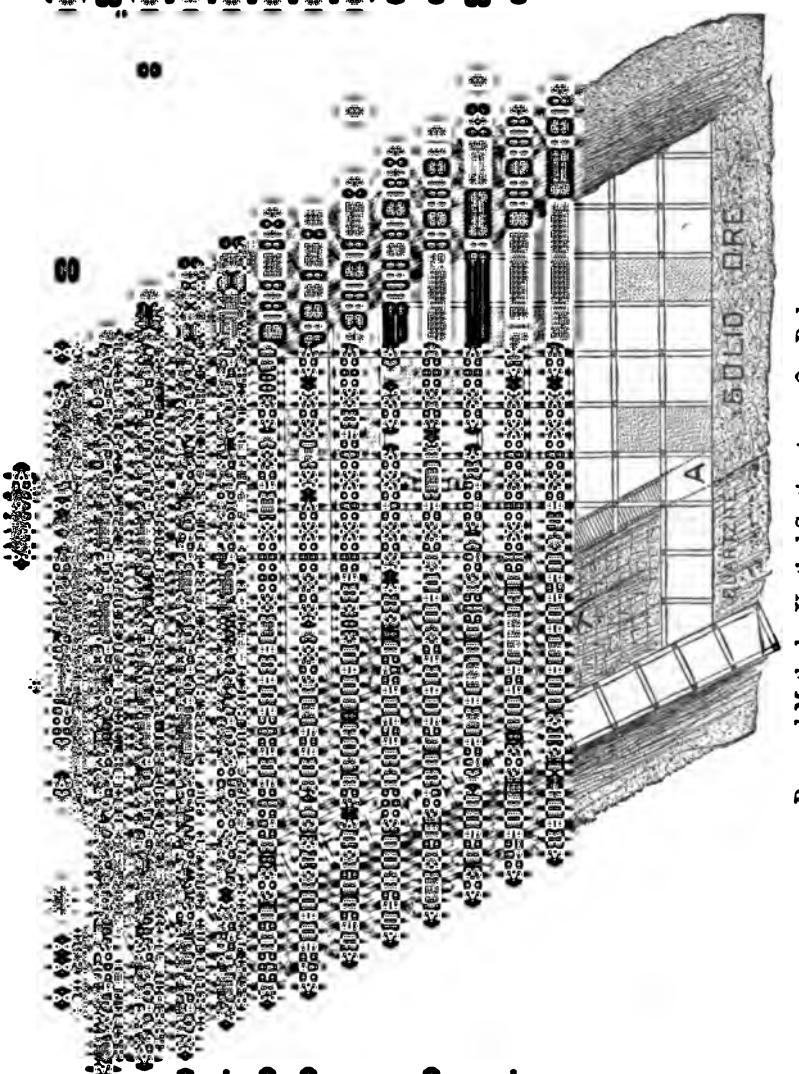

Proposed Method. Vertical Section Across Ore-Body.

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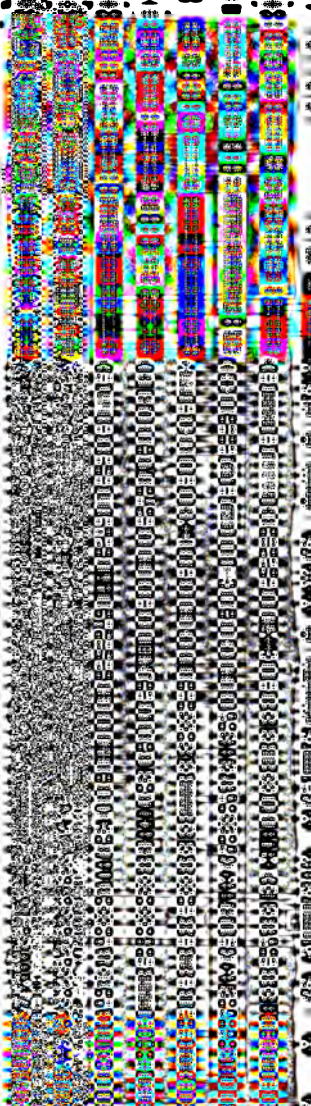

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te to hold the ore, which can
drawn from it into the cars
ow, or the mill or chute
ough which the ore is sent
wn can be built, say 3 feet 6
hes by 4 feet in the clear sec-
n of round hardwood sticks,
ed with hardwood plank, as in
Iron River mine, on the foot-
ll, as shown in the drawings.
a little care in packing around
this can probably be held in
packing as a waste "mill"
chute, through which waste
terial may be sent down from
surface, or, if it be found de-
able, the "mill" can be cut in
foot-wall, as indicated by the
ted lines in the drawing.

pper worked-out ground, lon-
two sets in height and one
distance between the cross-
, leaving intervals or pillars

between them of two sets in width, or more, if the ground will permit. These longitudinal stopes are timbered lightly, having to stand but a very short time, and being in the solid undisturbed ore, with the filling from above resting on each side of them on solid ore. When the mid-distance between the cross-stopes is reached the back-stopping commences by taking out the ore on each side *E, E, E* of the *C, C, C* stope, to the width of a set, or half the pillar left between the longitudinal stopes *C, C, C*, supporting the "gob" roof while doing so, and laying lagging poles or slabs across the floor of the stope as the work proceeds. Any waste rock or material available or desirable may be thrown back in the packing, and when a space *E, E, E*, the area of one or two sets, is worked out on each side of the last sets of the longitudinal stope *C*, the temporary timbering is drawn, and the "gob" roof, with the lagging previously laid under it, is allowed to drop on the bottom of the stope *C C C, E E E*, lagged to receive it. Light poles, and even brushwood, will serve for thus keeping the ore from mixing with the waste.

It will probably be found possible, as well as advantageous in many cases, to drive these longitudinal stopes *C C C* two sets high and draw back the upper one a little in advance of the lower, and as they can be driven out at any point in the cross-stope *B B*, as shown in Fig. 2, a whole horizontal slice or section of the ore-body, no matter what its thickness, can be opened out, say two sets in height at the same time.

It is not necessary to enter here into further details that must necessarily be subject to local conditions, and that can be laid out by the engineer in charge.

The advantages claimed for this system of mining are:

First. There is no dead work in it—all the work is simply stoping.

Second. All the levels and stopes that have to be timbered are in solid, undisturbed ore, and being only one "set" in width, are easily held and require but light timber, while much of this is drawn and saved in letting the roof down. When the stope comes up to the filling, this has only to be supported over one set of timber, while it rests on the solid ore on each side of this.

Third. The packing follows the ore down, and as long as this occurs, the cave on the surface can be constantly filled from "borrow-pits" much cheaper than by sending the filling below. If finally found desirable, the filling can be sent down, as in other cases, through "mills," etc.

Fourth. This system renders it possible to obtain practically all the ore in the vein, and to get it free from mixture with waste.

Fifth. Disastrous caves or crushes are absolutely impossible.

Sixth. There are more working-faces, and therefore more ore can be extracted from a given amount of ground in a given time, than by any other system.

It is believed that, as compared with the Chapin system, there will be a very large saving in dead-work, in timbering, in filling and in wages, the latter from working in safer ground. As it is simply impossible to continue the present method of mining in use on the Gogebic, there is no need of making any comparison of cost with that now in use there, though the proposed method would undoubtedly be much cheaper than that.

A special system of mechanical ventilation will have to be adopted in any case as the mines become extensive, and there is no difficulty in adapting a system of ventilation to this method of mining. When the ore is hard, little or no timbering would be required, and as the system is adapted to any kind of ore, there is no reason why, with proper modifications, it may not be applied in such mines as the Calumet and Hecla, where it would of course do away with the heavy timber-account, and render fires impossible.

A DESCRIPTION OF THE PLANT OF THE BOSTON HEATING COMPANY.

BY ARTHUR V. ABBOTT, C.E., NEW YORK CITY.

(Boston Meeting, February, 1888.)

In a few places attempts have been made to introduce some means of delivering heat from a central station. Probably Pittsburgh, through the advantages derived from its almost inexhaustible supply of natural gas, has made a more wide-spread success in this direction than any other city. From the gas-wells in the vicinity of that town an enormous supply of natural gas can readily be obtained at a pressure sufficient to force it many miles from its source and to distribute it to all consumers. Obviously, very few places have such natural advantages, and some other means must be devised, if it is desired to furnish heat in a location not supplied with gas-wells. Several plants have been introduced to deliver heat by

means of a number of boilers located at a central station, supplying live-steam to a series of pipes extending through the streets of the district to be served. The steam thus distributed may be used in any way in the same manner as if it was drawn directly from the boiler itself. Where plants of this kind have been introduced with appropriate engineering skill, and with due precautions against the liabilities to which they are exposed, steam-distribution has been successful. Recently another idea has been introduced, which, it is believed, will obviate some of the difficulties which attend the use of steam for the distribution of heat on a large scale and which will enable the necessary plant to be constructed much more cheaply. This system involves the circulation from the boilers of hot water instead of steam. The object of a heating-system is to *distribute* heat from place to place; and whatever means is used to carry heat from point to point is simply auxiliary, the distribution being the end to be accomplished.

The unit of heat adopted in this country is the quantity necessary to raise a pound of water one degree. Strictly, I should say that a unit of heat is the quantity of heat necessary to raise a pound of water from 38½° F. to 39½° F., the point of the maximum density. Water was selected as a medium for the measure of heat, because it was supposed to have the greatest capacity for heat of all known substances. It is now known that a solution of sugar and the element bromine have slightly greater specific heats; but this does not practically affect the convenience of using water for the purpose named, and saying that 1000 units of heat is the quantity necessary either to raise one pound of water 1000 degrees or 1000 pounds of water one degree. It is true, again, that the specific heat of water increases slightly as the temperature rises; but that increase is so very small that for anything but the most accurate and exact calculations it may be neglected.

To carry any substance from one point to another, we wish to select for our means of transportation that arrangement which will enable us to convey the greatest amount for the least expenditure. If we want to haul 1000 tons of earth, we get the largest cart that the horse can easily draw; if we wish to carry a load of rails, we obtain the largest car that can accommodate them. So, for the transportation of heat, we naturally select that substance which will convey the largest amount of heat. Mercury, oil, steam, hydrogen gas or petroleum could be employed; but, inasmuch as water, per unit of *volume*, will contain the most heat, it is obvious that it is best

adapted as a vehicle. It would be possible for us to use a solution of bromine or sugar, to which I have alluded, but these substances would have too slight an advantage in specific heat over that of water to render their use advisable.

Reference to the accompanying illustration (Fig. 1) may aid an understanding of our system.* At a central station a number of boilers are located, exemplified in the illustration at A. From the boiler A, proceeds a pipe C′, to pump B. This pipe is attached to the suction-end of the pump and consequently the action of the pump withdraws the water from the boiler. Proceeding from the discharge-end of the pump, a pipe C, extends through the streets, and, returning to the central station, enters the boiler at C″. As soon as the pump is set in operation, the water flows out of the boiler by the suction-pipe C, and is forced around through the streets and back again into the boiler by the pump. If, during its passage, no water is taken from the main, every stroke of the pump withdraws from the boiler, and returns to it again, an equal quantity of water. In reality the office of the pump B, is simply to sustain a continuous circulation through the hot-water main.

Directly beneath the hot-water main C, is a pipe D, which in the station terminates in the tank D′. This second main, called the return-main, collects the water as fast as it is used and cooled, and returns it to the station. As the return-main empties into the tank D′, all the water which has been cooled and carried back to the station is delivered into this tank, from which a second pump D″, draws the water and forces it back into the boiler, again to receive a fresh quantity of heat and be ready for another journey through the supply-main.

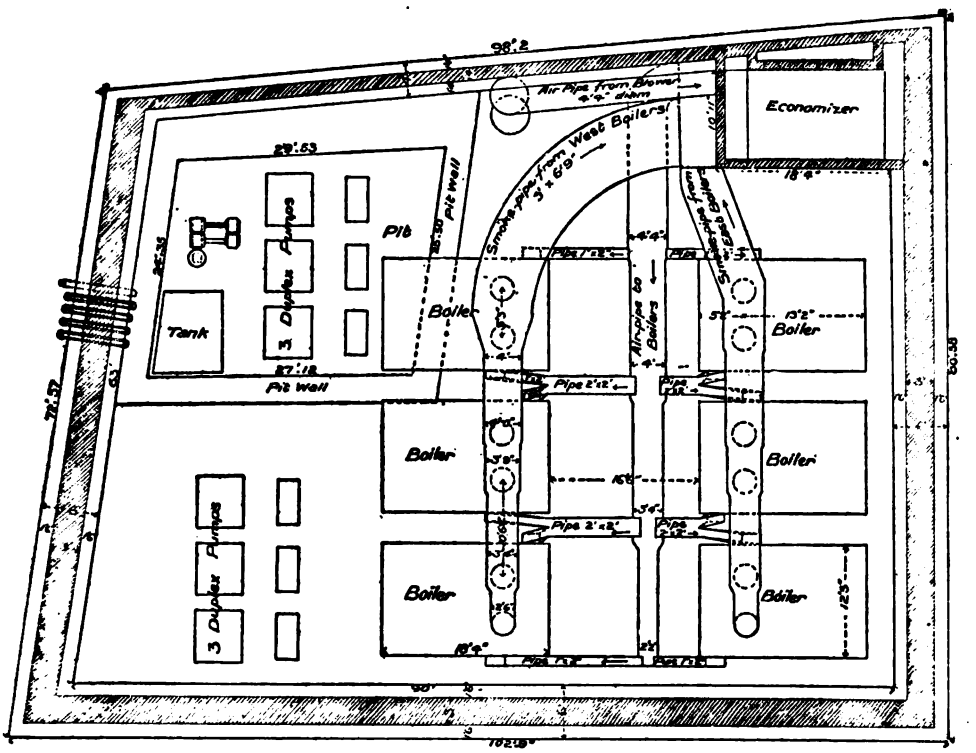
From point to point along the main, small pipes c', extend to the curbstone and terminate in the service-box N. The pipe in the service-box is so arranged as to enable a single box to supply three houses. This is accomplished by capping the end of the pipe with a three-way tee, to which are attached three asbestos cocks. From this tee in the service-box small copper pipes O, extend into the adjacent houses, furnishing them with a supply of water. Directly beneath the supply-pipe c'', is a similar, though a larger, pipe d', to collect the cooled water from the houses and conduct it to the return-main. This pipe d', also enters the service-box, and there by a similar ar-

* The actual plan of the station at Boston is shown in Fig. 1*a*.

rangement of tees and cocks, is enabled to receive the water from the three buildings which the hot-water pipe directly above supplies.

In the plant which we have been introducing this season the return-mains are all laid to grade, so as to conduct all the cooled water to the station by gravity. It is, however, possible to arrange the house-appliances so that they may be capable of returning the

FIG. 1a.



Plan of Boston Station.

water to the station over a higher grade than that at which they are located.

Neglecting any slight leaks unavoidable in so large a plant, and excluding waste that may occur from a thousand and one contingencies, the system once filled will always remain full, the water being simply the vehicle by means of which the heat received from the central station is transported to a distance. It is the car in which freight is carried, and after it has left its load of goods it returns to the station to receive another and repeat its journey.

In laying the mains a trench has been first excavated, some 2½ or 3 feet wide, and varying from 2½ to 7 or 8 feet in depth, having an average depth of 4 feet. The trench has been excavated to grade between the street corners. Along the bottom of the trench we have spread a uniform layer of concrete, 8 inches in thickness, made of 1 part of cement, 2 parts sand and 2 parts broken stone, thoroughly rammed into place. Once in about 15 feet a brick pier has been introduced into the concrete and solidly imbedded therein. On this brick pier has been placed an iron construction (Fig. 2) called a bracket.

The bracket consists of a solid, arch-shaped casting, supporting a roller covered by a cap. The office of this roller is to carry the 4-inch supply-pipe and allow it sufficient ease of motion so that it may readily expand and contract under the variations in temperature, while the cap surmounting the whole confines the pipe sufficiently in its place to maintain it in a fairly straight line and prevent it from becoming in any way displaced. The whole bracket stands on top of the brick pier, while directly underneath the arch of the bracket a second roller, placed on a small iron stand, is seen, the office of which is to support in a similar manner the 8-inch return-pipe, and to permit perfect freedom for its expansion. The difference in size between the supply and return-pipes is due to the different conditions of their operations. The supply-pumps at the station, taking their suction from the boiler, are able to maintain through the small supply-pipe a rapid current. We expect to carry a circulation, depending upon the demand on the system, of from 5 to 10 feet a second. By means of the pumps this forced circulation is easily accomplished. While a small pipe for the supply adds to the frictional resistance offered to the pumps, the radiating surface is largely diminished, the cost of the pipe is much decreased and the ease of construction is greatly facilitated. In the return-pipe, when the water is to come back to the station by gravity alone, it is necessary to decrease the frictional resistance as much as possible, to afford an abundant chance for the water to run back easily and freely, no matter whether the discharge from the houses is regular or irregular. So we have for the return an 8-inch and for the supply a 4-inch pipe.

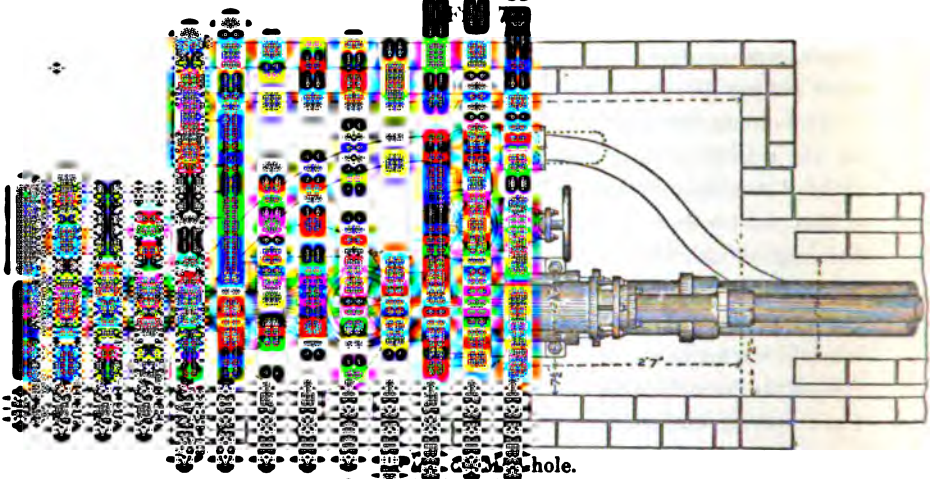
The territory covered by our plant is shown on the map (Fig. 3), while the construction of the conduit is illustrated by the longitudinal section (Fig. 4) and transverse section (Fig. 5). In actual steam-practice it is found absolutely essential, wherever there occurs

any change in direction of the pipe-line, to introduce some means to provide for the expansion which is due to the variation between the temperature at which the pipe is laid and that which it attains as soon as circulation takes place. Experiments on various mechanical contrivances have convinced us that the best joint to be used for our purpose is the telescope expansion-joint.

The expansion-joint (Fig. 6, elevation, and Fig. 7, sections) consists of a large casting having two longitudinal holes into which the ends of the supply- and return-mains are introduced. At one end of the casting these holes are supplied with threads E'' (Fig. 7), and the ends of the pipe are screwed into them in the same way as they would be introduced into an ordinary coupling. By this means, as the expansion-joint is firmly bolted to the masonry foundation, the joint forms an anchorage, so that one end of the section of pipe to which the joint is attached is firmly fixed and held in its place. At the opposite end of the joint from the screw-threads the casting is enlarged to receive two sleeves K, of phosphor-bronze, containing a large percentage of aluminum. These sleeves form the movable part of the joint, and after being introduced into the casting, are carefully packed with a rope J, made of asbestos fiber, impregnated with black lead. This packing is introduced between the casting and the phosphor-bronze sleeve in the same manner as ordinary packing is introduced into stuffing-boxes; and the gland L, is firmly fastened on top of the packing. The outside end of the phosphor-bronze sleeve is furnished with a screw-thread C, to which the end of the pipe is attached, so that when the pipe expands or contracts, the phosphor-bronze sleeve moves in and out of the casting and accommodates itself to the varying lengths of the pipe. By means of the gland in the stuffing-box and a corresponding ring e'', on the inside of the casting, the phosphor-bronze sleeve is very carefully aligned, so that its motion in and out is in a straight line. In the joints, which we have introduced here, the sleeve of the supply-main is long enough to give a motion of 12 inches, while that of the return-main is about 8 inches. Inasmuch as these joints are placed, on an average, as often as once in 100 or 150 feet, and the maximum motion for which they will have to provide is only from 4 to 6 inches, it will be seen that there is an abundant margin to prevent any possible cramping.

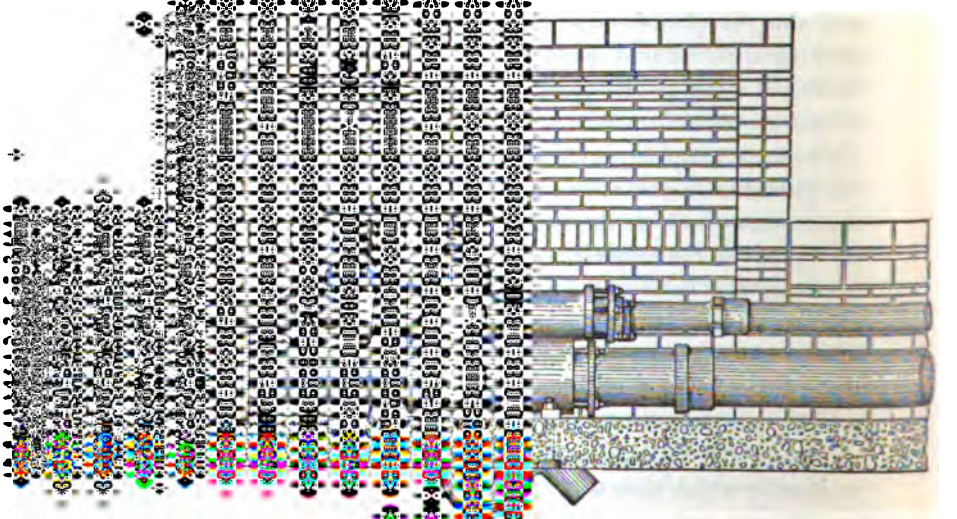
Last spring we built an experimental joint of this kind and setting it up in our shop in New York, put on a steam-pressure of 450 pounds to the square inch, and attaching a lever to the sleeve

[illegible] thousand times, corresponding to [illegible] contractions of the pipe. At the



[illegible]-hole.

[illegible] the beginning, not leaking a



[illegible]-hole.

[illegible] placed in a man-hole (Fig. [illegible] it is perfectly accessible to [illegible] end of the expansion-joint

there is a valve. The object of this valve is two-fold. Beyond the valve, in the casting of the expansion-joint is a side-outlet, provided with a valve. In the growth of the system it will soon be necessary to introduce cross-pipes, extending between the main supply-pipes, passing through the side streets, so as to give a hot-water supply to the intervening buildings. For example, there is a man-hole at the corner of Devonshire and Franklin streets and one at the corner of Franklin and Congress. At each of these man-holes occurs an expansion-joint. At any time it is simply necessary to connect this valve at Congress street with the corresponding one at Devonshire street, and then, opening the valves, a stream of hot water would flow between the two streets, making a cross-connection from which the buildings on those streets could be supplied.

When it becomes necessary to repack the expansion-joints (though, to the best of our belief, the packing will last a long time), it is only necessary to shut off the valve at one man-hole and a corresponding valve at the next, to cut a section of the main out of the circuit, and by opening a side-valve we can discharge the water contained in the main into the conduit (which is provided with a drain for this purpose), and then, by blowing a stream of air into the man-hole, cool it off sufficiently, so that the workman can open the gland and introduce a new packing—all in the course of a few hours. It could easily be done at night, when the demand for heat was a minimum.

Many questions have been asked as to the safety of this system, and exaggerated statements have been current as to the pressure which we propose to carry.

The supply-pipe is made of what is called "extra heavy" pipe, the bursting-strain of which is 12,000 pounds to the square inch, as we have ascertained by testing a number of samples to destruction. Every piece of pipe that has gone into the streets has been tested to 4000 pounds to the square inch as a proof-test. After the main was laid in place every section (that is, the distance from one expansion-joint to the next, including all screw-threads, all of the packing of the expansion-joints, and all joints) has been tested to 1500 pounds, and after the main was completed, the whole was again tested up to 1500 pounds, from the station round back to the station again. So, when the main was completed and ready for use, it had received, *first*, a test at the mill of 4000 pounds; *second*, a test by sections of

1500 pounds; and *third*, a test, as a whole, of 1500 pounds—all being pressures per square inch.

We expect to deliver water to our customers at a temperature of about 400 degrees, which corresponds to 250 pounds to the square inch absolute, or 235 above the atmosphere. The water will have to leave the station slightly higher than that, to provide for the unavoidable radiation. If we should send the water from the station at 300 pounds to the square inch and allow 100 pounds for pumping friction, the total pressure on the main would not exceed 400 pounds, and as the main has been tested to 4000 pounds, there is a large margin of safety.

As an additional precaution we have once in every section, a check-valve, so arranged as to shut off automatically each section of the main in case of an accident. Malicious injuries might occur, for it is conceivable that in times of strikes men might dig up the main or drive a pick into it. In the illustration (Fig. 8) a cross-section of this valve is shown in connection with the expansion-joint. It will be seen that on the left-hand side of the joint the casting is enlarged, so as to form a spherical cavity, into which one end of the pipe C, is screwed in the same manner as into a regular coupling. The spherical cavity contains a ball, supported on two ribs, so planned as to allow the ball when at rest to remain at the bottom of the spherical cavity. The end of the pipe C, is chamfered so as to form a valve-seat. Under ordinary circumstances the ball remains at rest on the ribs. Should, however, any rupture occur, the current of water flowing through the main will, by reason of the break, be greatly accelerated and acting on the ball will cause it to roll up along the ribs and seat itself on either side of the spherical cavity toward which the current may be flowing, thus completely shutting off the remainder of the main. The forces keeping the ball in equilibrium are its weight, acting downwards and keeping it in place and the friction of the water-current tending to force it up the inclined ribs and seat itself against either of the sides of the spherical cavity. Hence, by varying the weight of the ball, the valve can be adjusted so as to close with a given velocity of current. Under maximum demands we can use a current of 10 or 15 feet a second, and the ball is so weighted as to close at a velocity of 20 feet a second. Should a rupture in the pipe occur, giving a velocity of 20 feet a second, the ball will leave its place, and rushing up, close the end of the pipe and shut off the rest of the main. This is not simply theory, but practice, to the extent that we have made a number of

these valves, and, after experiment, have found them to work very accurately.

Should any injury, either malicious or otherwise, rupture the main, it is obvious that only the quantity of water contained between two check-valves would escape from such a break. As these valves are placed at intervals of about 100 feet, the amount escaping would not exceed 20 cubic feet. The volume of the conduit is so large that, should this entire quantity of water be discharged into it, the steam formed therefrom would be quickly dissipated through the length of the conduit without producing sufficient pressure to do any damage.

The conditions which surround a pipe in the street are so different from those to which boilers are subjected that a little consideration will show an explosion of the pipe to be an impossibility. A boiler, with its setting of masonry and bed of incandescent coal, is encompassed with a highly heated atmosphere which constantly tends to supply it with more and more heat. The street pipe, on the other hand, is hotter than its surroundings. On the occurrence of a slight rupture or tear in the shell of a boiler the pressure is relieved from the large mass of water therein contained, and an outflow of the boiler-contents is established through the incipient opening.

The large diameter of the boiler-shell permits the molecules of water flowing towards the incipient rupture to attain, before reaching it, a very high velocity; while the hot masonry surroundings, and especially the glowing coals and incandescent gases of the furnace, furnish to the water continuous supplies of heat, maintaining the pressure and accelerating the rushing molecules. So, in far less time than it has taken to describe this action, the current of steam and water has attained such velocity that its impact has been sufficient to rend the boiler, and perhaps overthrowing the masonry, hurl it hundreds of feet from its original location.

In the street-pipe the comparatively small diameter precludes the possibility of a high velocity in the water-current, even should an opening occur. Furthermore, as no supply of heat is furnished to the water, the pipe being surrounded by the comparatively cold conduit, every unit of steam formed abstracts and renders latent about five units of water. Thus, even if a rupture occurred in the pipe, no disastrous explosive action would follow, a simple tear through which the water would escape into the conduit being the only result.

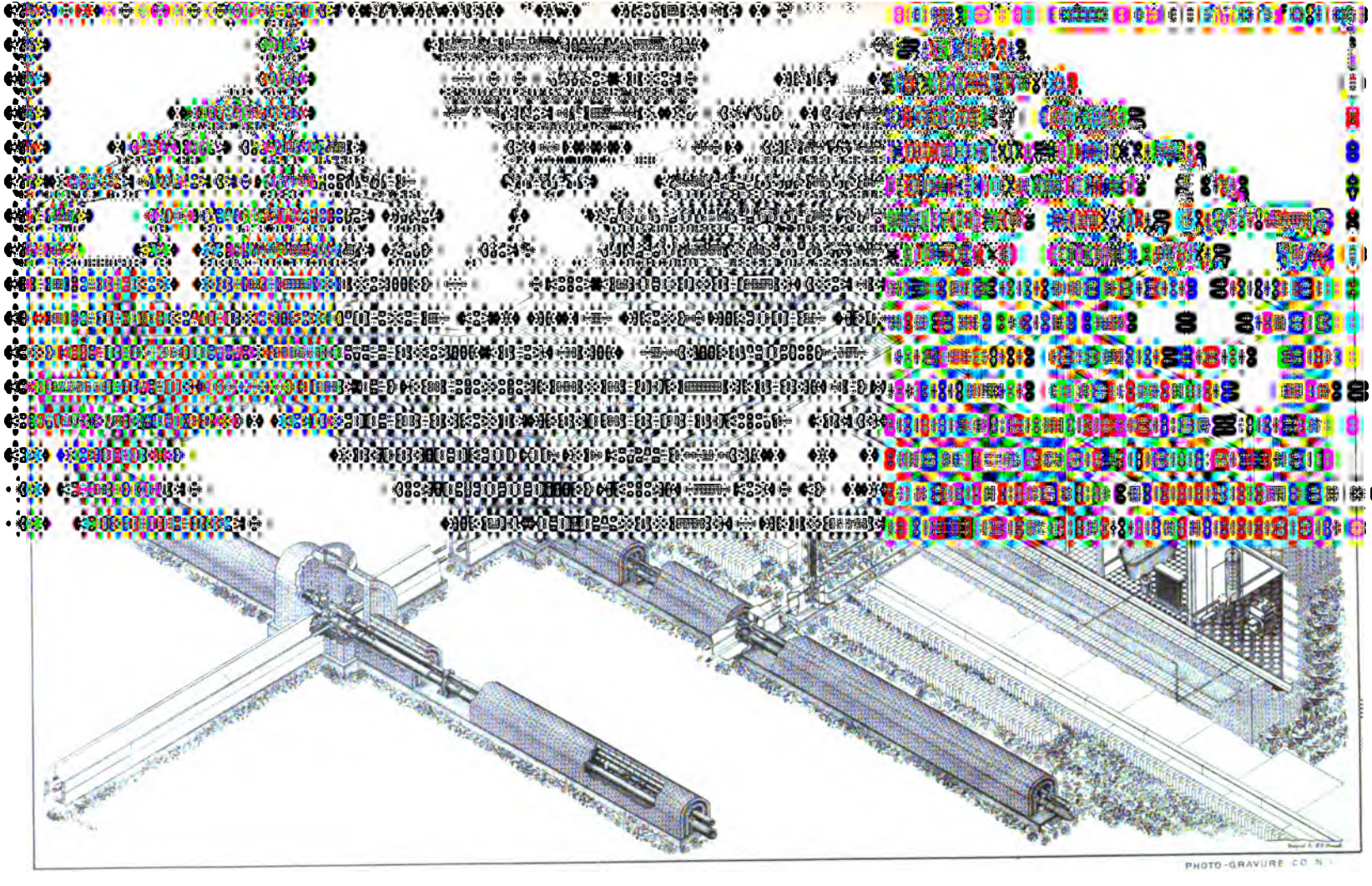
Before passing to the house-connections between the main and

the buildings, I would call attention to the special screw-thread which we have used in making the joints in the streets, with the two-fold object of securing extra strength and greater tightness.

Ordinarily a screw-thread, as is well-known, reduces the strength of the pipe or rod on which it is cut about 30 per cent. In Fig. 9 the special thread used in our plant is exemplified. The coupling joining the ends of the pipes B and C is made considerably longer than is customary in ordinary pipe-fittings. For a little way, the end of the coupling is bored out so as to be a fairly accurate fit on the end of the pipe. This greatly improves the joint, as the overlapping end of the coupling tends to strengthen and support the pipe that is introduced into it.

The special peculiarity of the thread, however, is that portion between the points bb' and cc'. It will be seen that the top of the thread is in a straight line with the outside of the pipe, while the bottom of the thread, between the points bb' and cc', is inclined to the axis of the pipe at a considerable angle, so as to cause it to run out or vanish at b' or c'. By this means, the weakening of the pipe, caused by the cutting of the thread is spread out or diffused over a considerable length, and by proportioning this vanishing of the thread in a proper manner, experiment has shown that it has been possible to preserve 97 per cent. of the full strength of the pipe. In addition, this vanishing of the thread produces a long and tapering cone which may be forced into the coupling by means of the pipe-tongs in such a way as to actually bed the metal of the pipe into that of the surrounding coupling and make a joint which is absolutely tight. The same result is attained in a less degree with the ordinary thread; but, inasmuch as the cone produced by our special thread has a very much smaller angle than that used in standard pipe-fittings, the pressure of the tongs in making up the joint causes it to bed more firmly into the metal of the coupling. The rolling-mill supplies pipe in lengths of about 20 feet. Hence the necessity of securing a perfectly tight connection between each two lengths is apparent. With this form of thread our experience has demonstrated that, even under 1500 pounds, it is perfectly possible to secure absolutely tight joints. In testing the sections we have never found a leak when the joints were properly made up.

Each coupling forms also an opportunity for a house-connection. On either side of the coupling a boss is cast. For the house supply inch-pipe is used, and for the return, two-inch pipe, which extend from the main to the sidewalk on either side of the street passing



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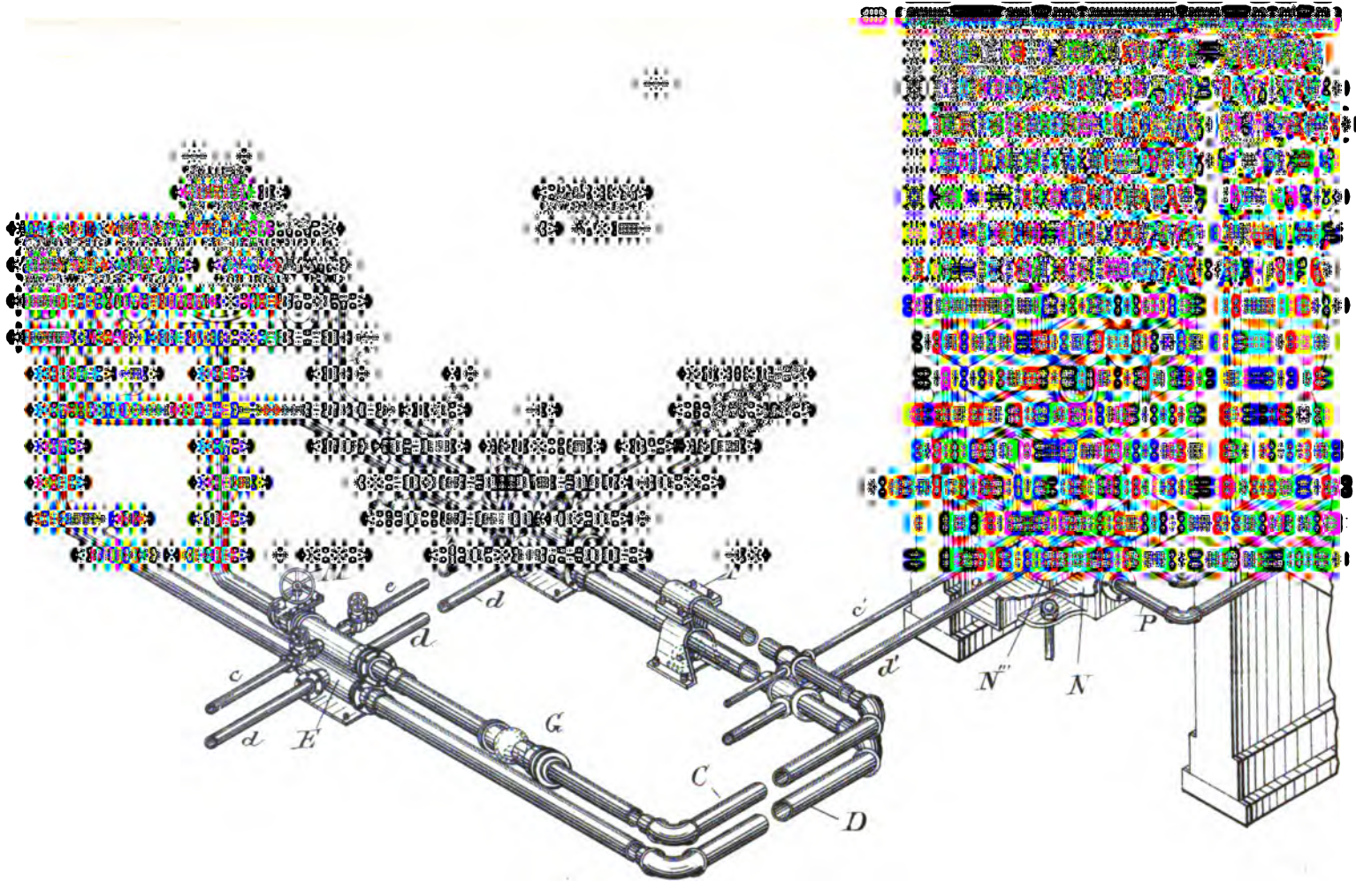
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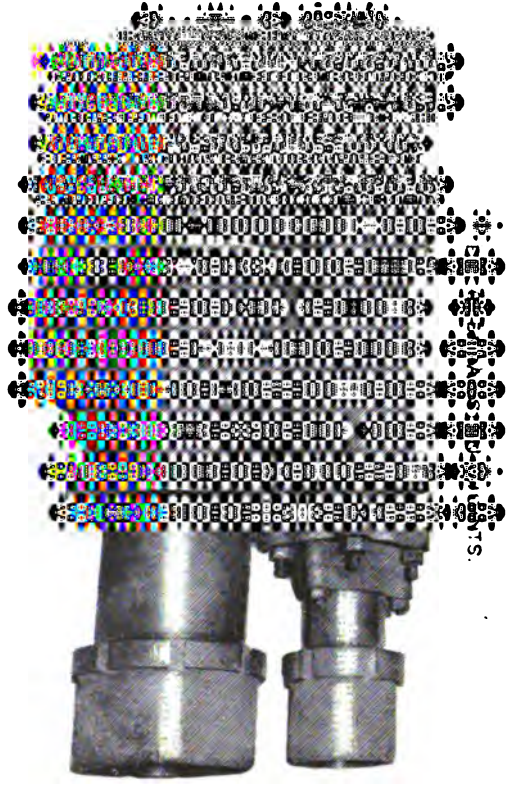

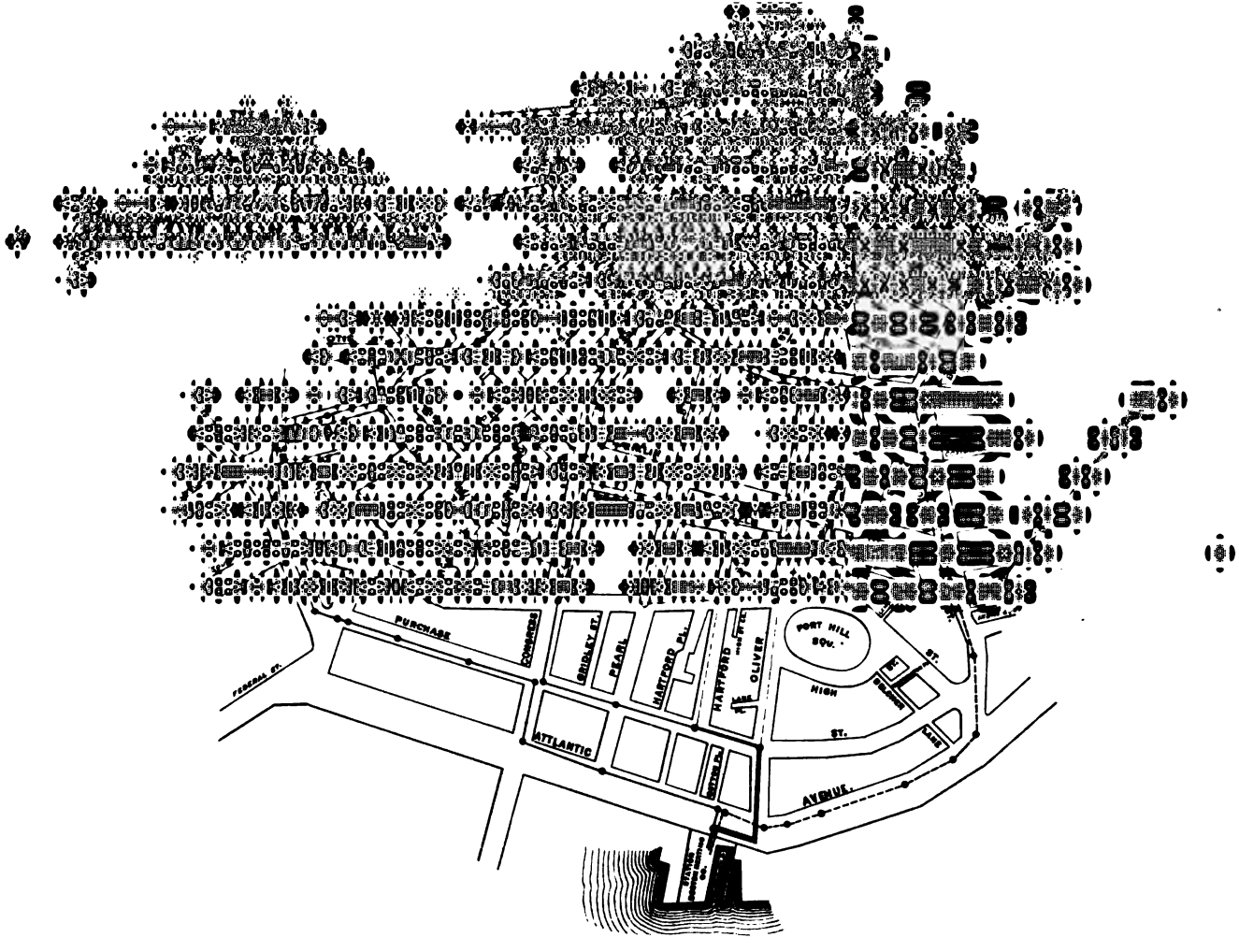
PURCHASE
CONGRESS
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PEARL
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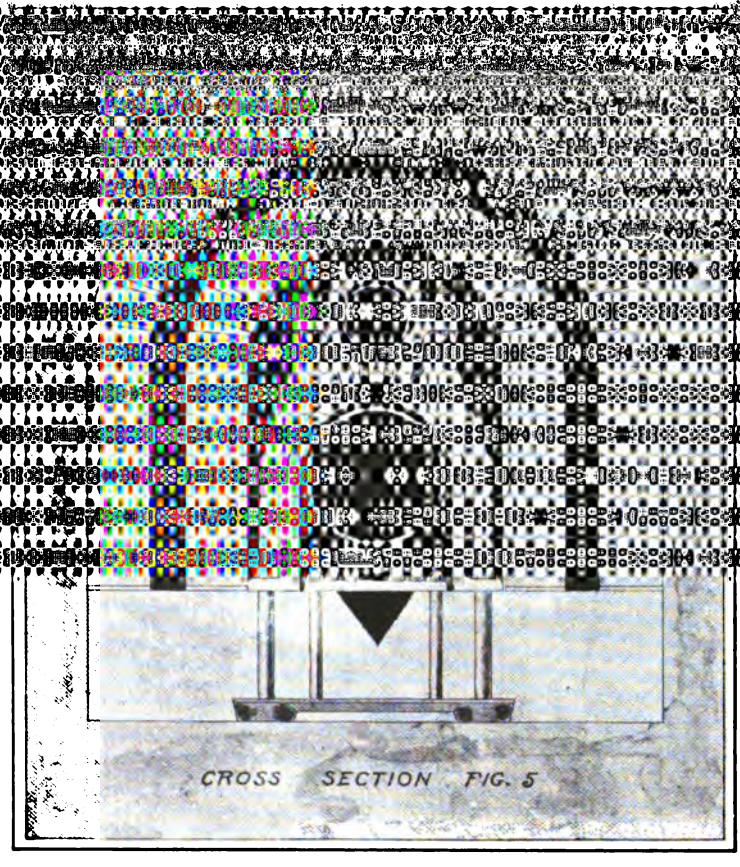
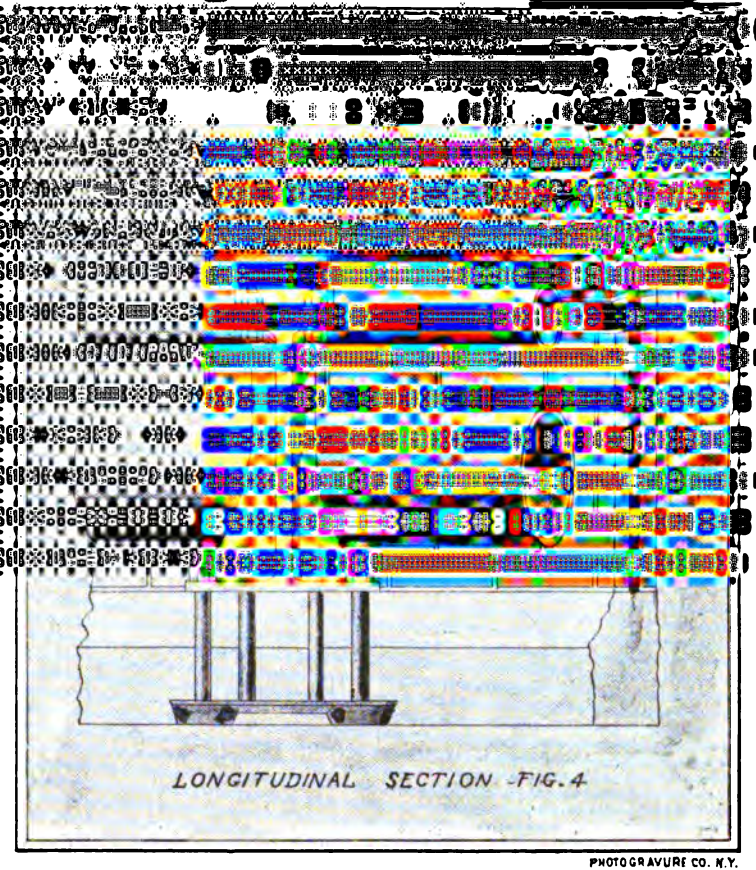
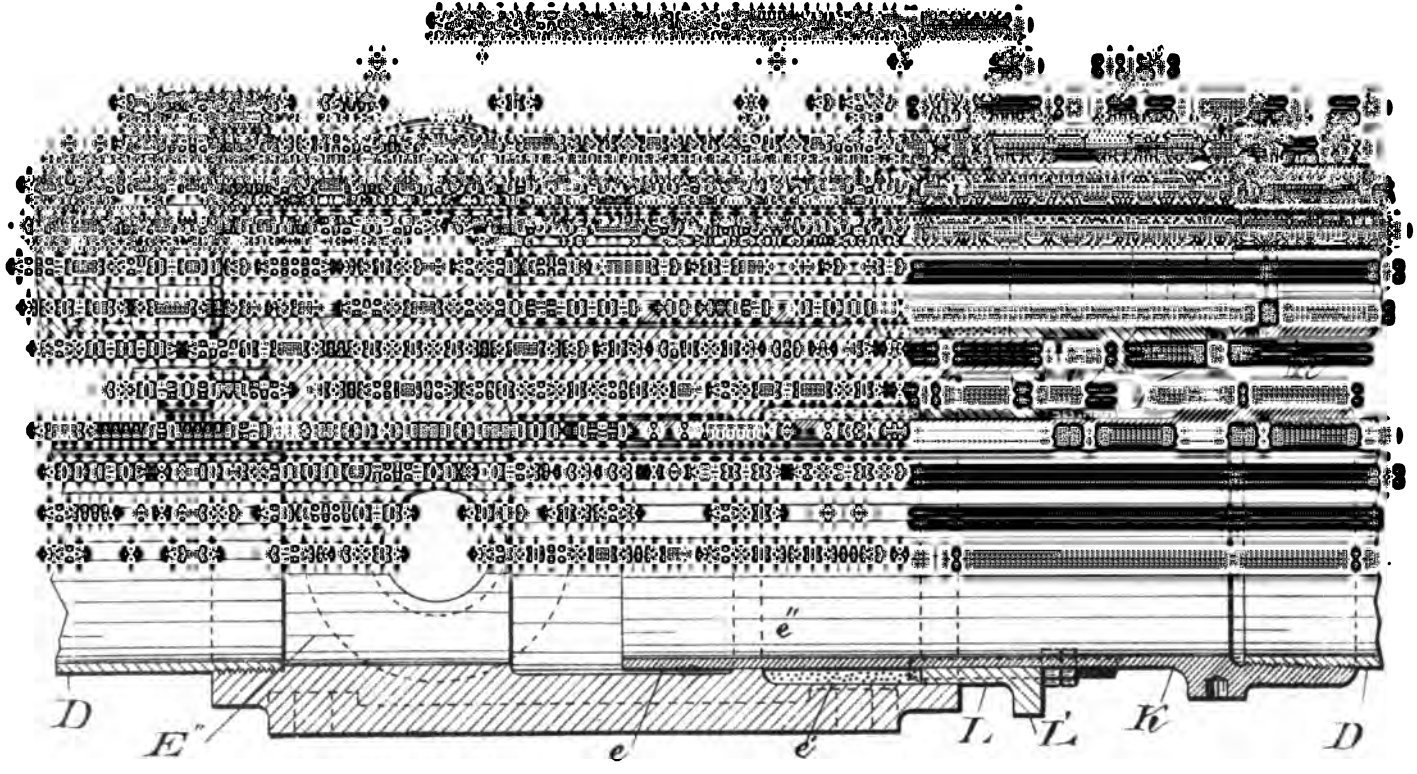


CROSS SECTION FIG. 5



LONGITUDINAL SECTION FIG. 4

PHOTOGRAVURE CO. N.Y.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D
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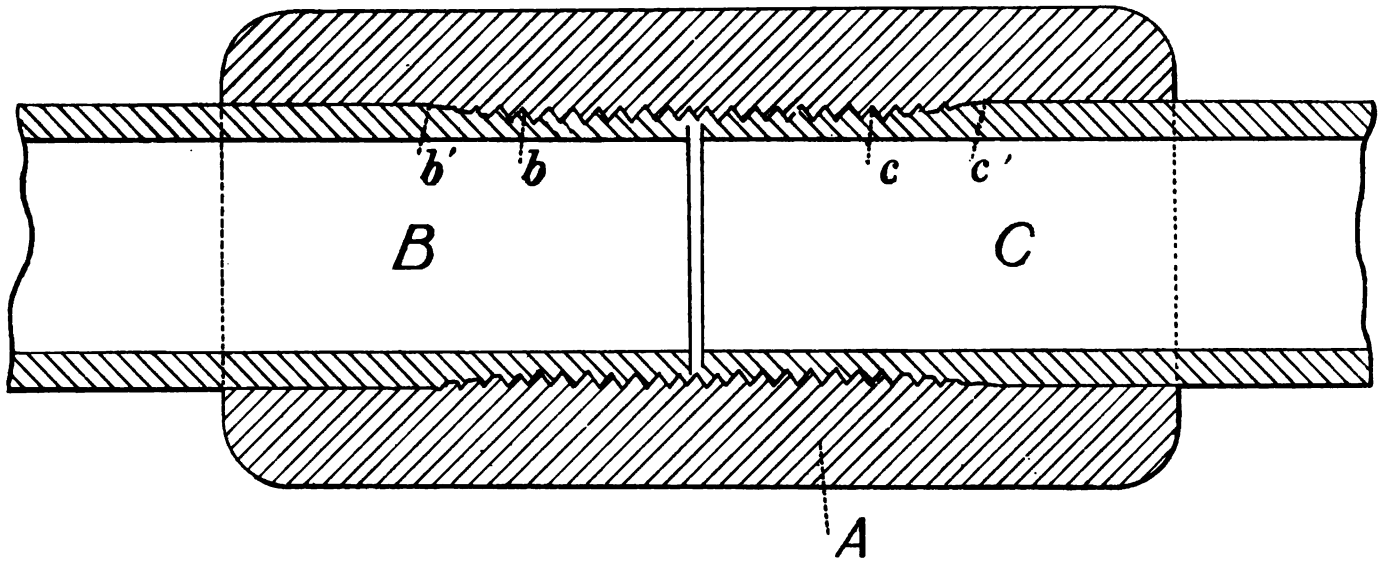
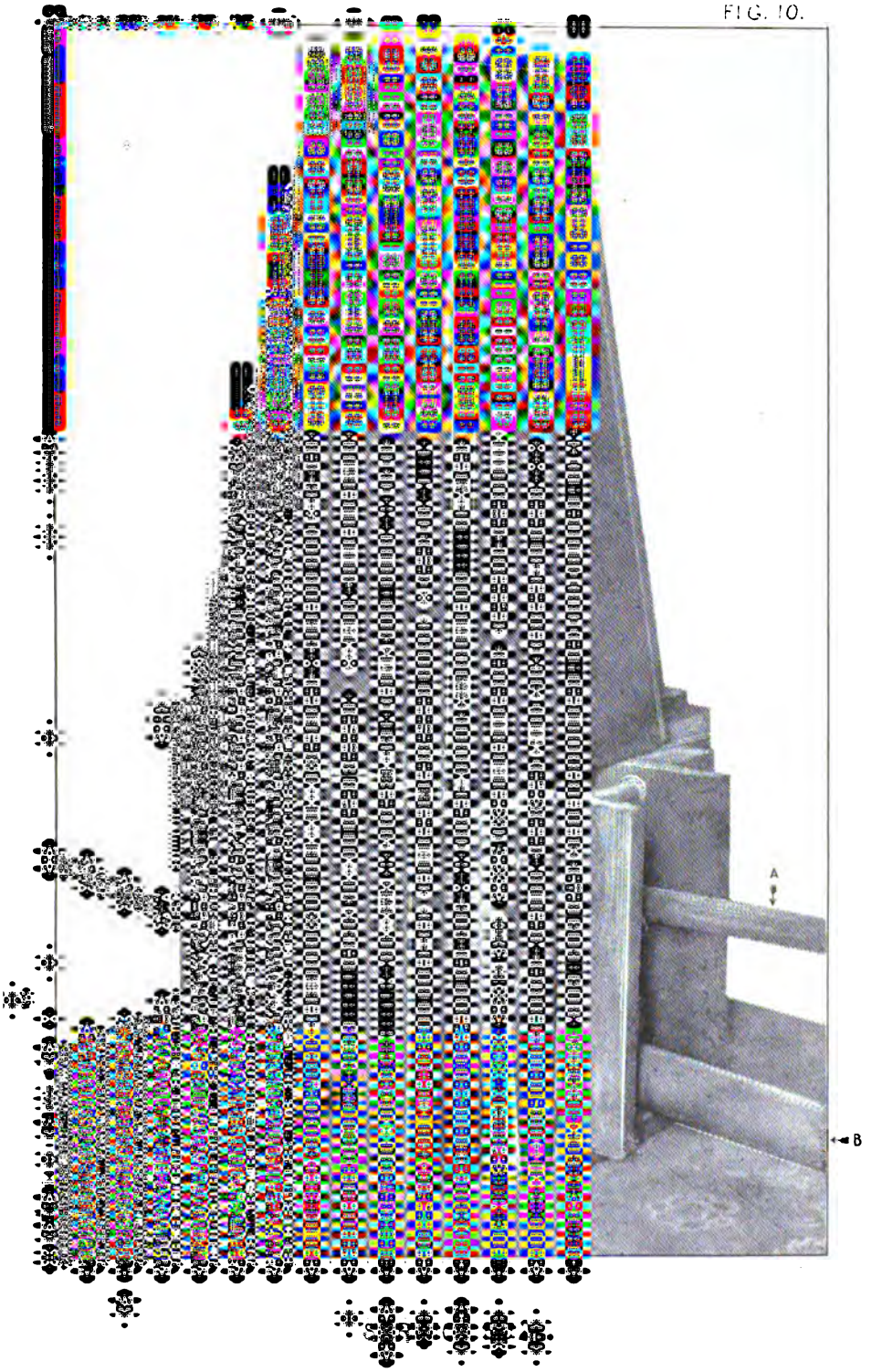


FIG. 9. SPECIAL THREAD.

FIG. 10.



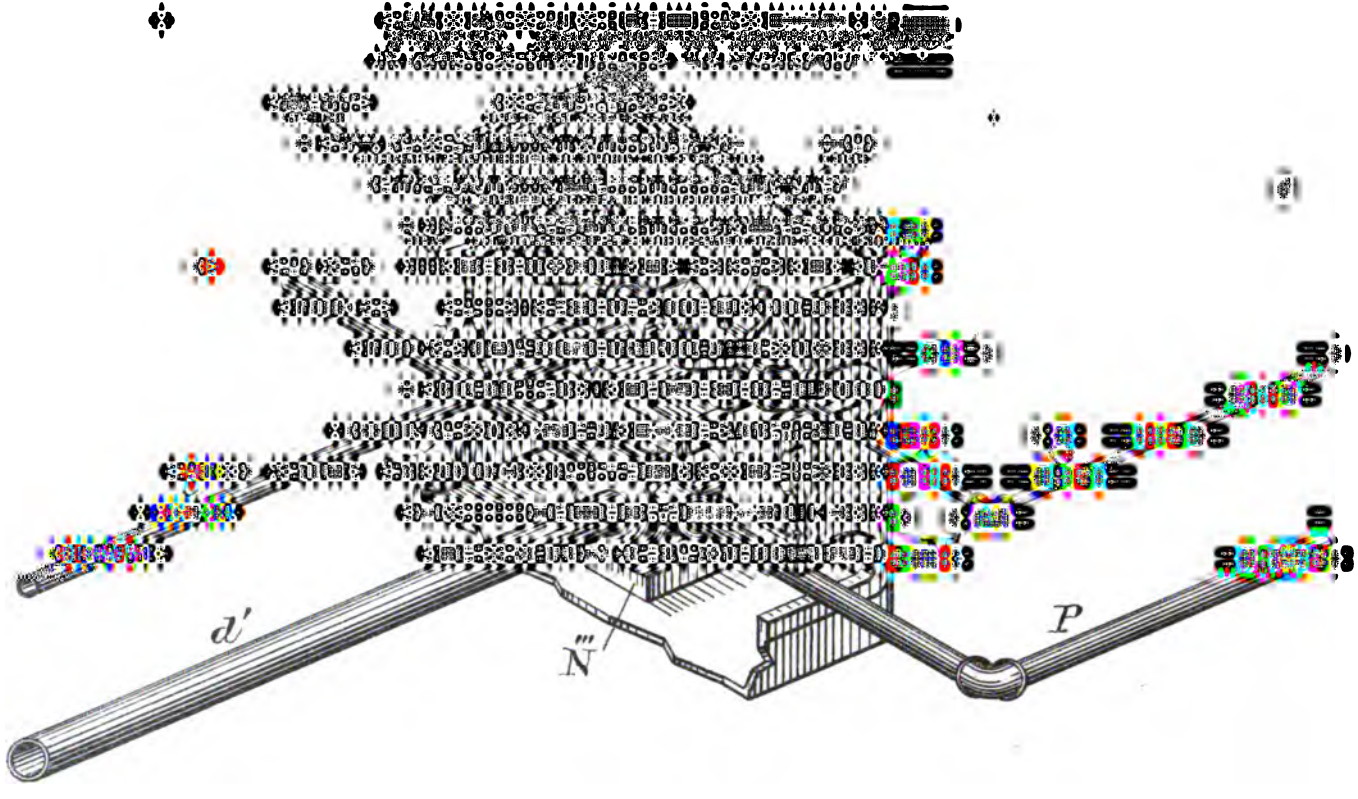
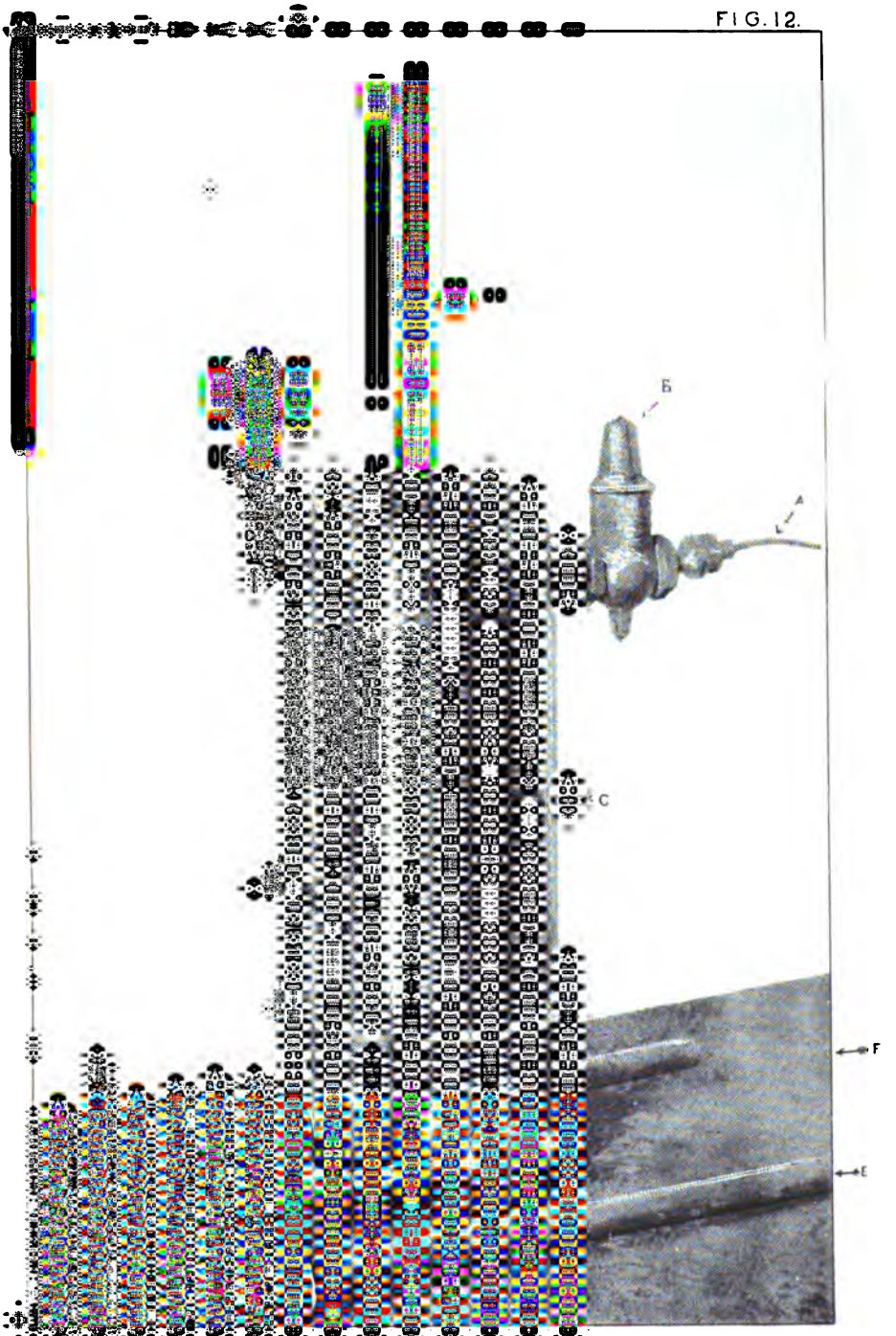
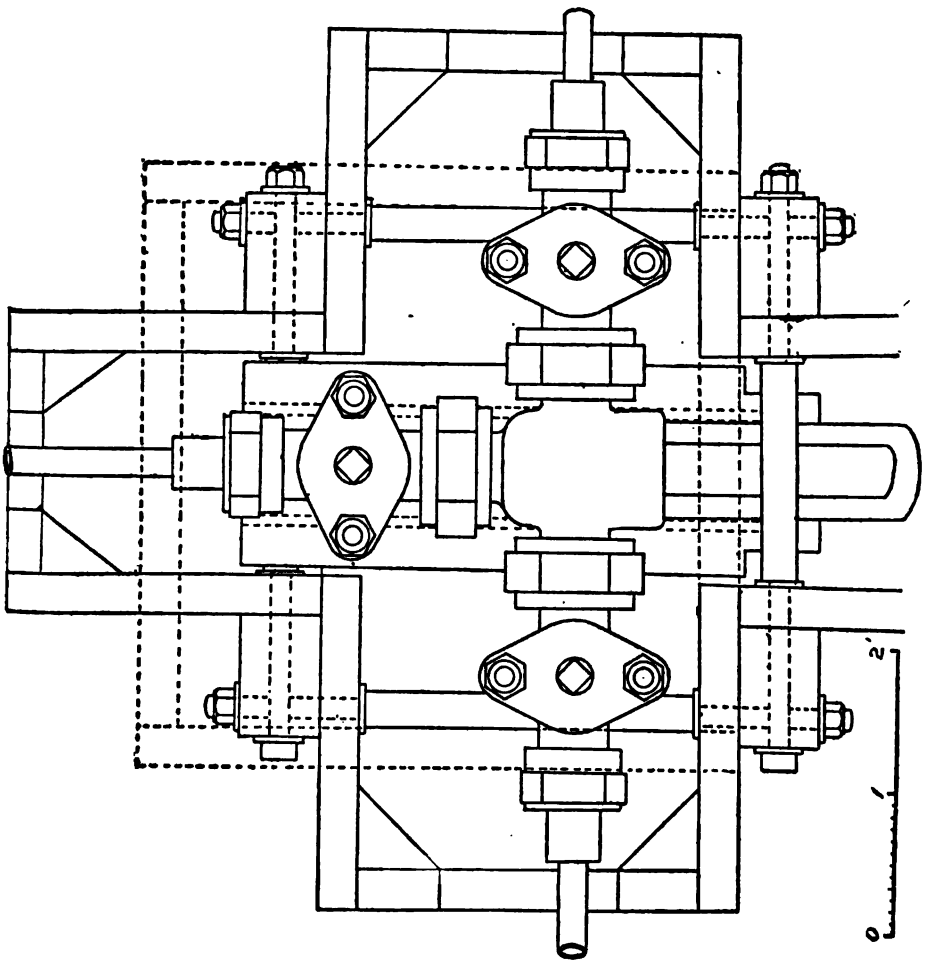

d'
N'''
P

FIG. 12.



through a box made of creosoted yellow pine. At the sidewalk a service-box (shown in elevation, Fig. 10, in section, Fig. 11, and in plan, Fig. 11*a*) is situated. In Fig. 10 the supply-pipe may be seen at A, while the return-pipe is indicated at B. These pipes, A and B,

FIG. 11*a*.



Plan of House-Connection.

enter the box and there terminate in a three-way tee, provided with asbestos cocks, by means of which the supply from either branch of the tee can be at pleasure controlled. By means of this three-way tee and its asbestos cocks, each service-box is enabled to supply three houses. From the service-box to the inside of the house-wall—

usually a distance of not more than eight feet—copper pipe is employed in preference to iron pipe. The advantage of copper pipe in this location is very obvious, when it is considered that owing to the ductility of this metal the pipe can be bent to any desired shape without the necessity of special fittings, involving the construction and maintenance of a large number of joints. By using from the service-box to the inside of the house-wall a copper-pipe, we are enabled to introduce in it without difficulty as many bends and carry it around as many corners as may be necessary.

The size of copper-pipe which we most frequently use in house-connection is quarter-inch, which is amply sufficient to supply ordinary buildings. In the case of large stores or warehouses three-eighths or one-half inch, and where it is desired to supply power to an engine of 25 or more horse-power five-eighths or three-fourths inch pipe is employed. A one-inch pipe would be ample to supply a building as large as the Boston post-office. All of these copper pipes have been tested to 6000 pounds. The sample of one-inch pipe so tested split at 6200 pounds, while the smaller sizes held 7000 without showing any signs of failure.

As to the use of the heat thus conveyed into the houses it may be stated broadly that our service is adequate to afford a supply of heat for any purpose whatsoever requiring a temperature of 400 degrees or less, whether it be for heating power, cooking, chemical operations or any branch of manufacturing. The appliances by means of which the heat contained in the water may be utilized are as varied as the different branches to which it may be applied.

For heating simply two plans present themselves. Hot water can be introduced directly into a radiator which may occupy the same position as the present furnace in the house takes up, and may warm a quantity of cold-air supplied through the cold-air box, and send that air heated through the flues that are already in place, so as to warm the building in the same way that the present furnace does, only substituting a hot-water coil for the glowing mass of incandescent coal.

Where the edifice is already piped for steam, or in case of a set of offices where a very varied supply is desired, steam-heating in the usual manner may be resorted to by the introduction of a device called a "converter." This contrivance, shown in Fig. 12, may be briefly described as a steam-dome, for in reality in our system it occupies the same place that a steam-dome does in a boiler. If, in imagination, you will conceive an ordinary boiler to be stretched

out so as to occupy two miles of space, you will have a very fair conception of our system as applied to the distribution of steam heat.

The end of the copper pipe A, as it comes in from the street is attached to a reducing-valve B. This reducing-valve is of rather peculiar construction, specially adapted to withstand the pressure to which it is subjected, and also to afford a regulator of unusual sensitiveness and durability. By means of the pressure reducing-valve most of the pressure on the water contained in the copper pipe is removed and the water allowed to flow into a large iron receptacle C, which forms the steam-dome proper. By the removal of the pressure, a part of the water is thereby permitted to take up the superfluous heat and to expand into steam.

On the left-hand side of the converter a small steam-gauge is shown, the purpose of which is to constantly record the pressure to which the converter is subjected, and to enable the reducing-valve to be set so as to give a pressure of any desired amount. In the top of the converter a steam-pipe B, conveys the steam away as fast as it is formed, and carries it to any part of the building where its use may be desired. At the bottom of the converter a return-pipe E, may be connected to a float-trap placed on the inside of the converter. Another pipe F, is used to convey back to the converter the condensed water from all the radiators, so that there may be little or no loss in the system. As fast as this condensed water accumulates in the converter, the trap, previously alluded to, discharges the water from the return-main E, and allows it to flow into the main in the street, whence it is conducted to the station.

As a precautionary measure a safety-valve G is attached to the converter, so that in case of any failure of the reducing-valve to act in a proper manner, which might possibly allow a greater pressure to come on the converter than is intended, this safety-valve will open and permit the contents of the converter to flow into the return-main, and relieve the former entirely.

For supplying steam to an engine no change is made in the converter, excepting to enlarge it sufficiently, so that there may be a quantity of steam always on hand ready to supply the cylinder of the engine. We generally calculate that, to preserve an adequate supply, it would be necessary to have the volume of the converter at least ten times that of the cylinder of the engine which it is desired to feed. So, for a large engine, we merely increase the size of the iron steam-dome to such proportions as shall always preserve

the requisite amount. For any cases where both heat and power are desired in the same building, as frequently occurs, we use a compound converter with two reducing-valves, so arranged that the water first introduced from the street shall expand into one chamber, giving, for example, a pressure of sixty pounds of steam for the purpose of driving an engine. As soon as the water, under the pressure of sixty pounds, is discharged from this first chamber in the converter, by means of the trap, it is received in a second one, where, by means of an additional reducing-valve, the pressure is again reduced, and the remaining portion of heat contained in the water allowed to expand a part of it into steam, which may be used for heating. By this means we are enabled to reduce the temperature of the water to the greatest amount, thereby returning it to the station as cool as possible.

In a system of this kind protection from radiation is an exceedingly important consideration. After a number of exhaustive experiments on nearly all of the non-conducting coverings now in use we decided to adopt a covering made of asbestos. The covering is simply a roll of pure asbestos fibre one and one-half inches thick. It is made by taking the asbestos from the mines, carding it in the same way that cotton wool is carded, and winding it around a cylindrical roll. After the mixture is dry a saw is run along the side of the roll, cutting the covering in two; then the roll is opened, and it is taken off. On the outside of the asbestos is a solidly woven cloth made of asbestos, rendered water-proof by an admixture of plaster-of-paris, and held in place by wire netting.

As shown in the section of the conduit (Fig. 5) we have, in the center, the pipe itself; outside of the pipe an inch and a half of asbestos with a water-proof asbestos covering. An air-space of four inches separates the asbestos from the first brick arch; then a second air-space of two inches, and a second brick arch. So we think the system is about as thoroughly protected from radiation as it could be. As to the insulating power of the asbestos the following experiment may be interesting:

I had an air-bath made, so arranged that it could be kept at a constant temperature of 500° Fahr. A sheet of the asbestos covering was laid on top of the air-bath. In contact with the upper side of the asbestos a piece of two-inch yellow pine plank was placed so as to cover the sheet entirely. Between the asbestos and the plank a second registering thermometer was introduced, so that the temperature between the asbestos and the plank could be accurately

ascertained. The experiment was continued for several days, during which time the air-bath was constantly maintained at a temperature of 500°, and the highest temperature ascertained as occurring between the asbestos and the wood was 158°.

The relative cost of transporting heat from point to point is a most important consideration. If we have a vessel in which we wish to maintain a constant temperature, whether it be a radiator or steam-heating, or a cook-stove, or steam-engine, it is necessary to supply the heating-medium to that vessel at a higher temperature than that at which it is to be sustained; and the greatest economy of maintenance is only achieved by supplying the medium to the vessel at the highest possible temperature and exhausting it therefrom at the lowest. In other words, to furnish the least quantity of the circulating medium with the greatest possible fall in temperature. If we supply a pound of water, for instance, at 400°, and let it cool down to 200°, we get 200 units of heat; if we supply it at 300, and allow it to cool down to 200, we get only 100 units of heat. So that in the practical operation of a system of this kind the aim should be to introduce the circulating medium at the highest temperature and reduce it to the lowest. The temperature required for cooking is about 350 degrees; and it is probable that this demand is the most severe that can be made on the system. Hence, for a discussion of the relative advantages of water over steam as a medium for the transmission of heat, I have selected this as being the one that would present the system in its worst light.

If a range is to be maintained at a temperature of 350 degrees it is proposed to supply water at 400 degrees. Suppose there is introduced into the range a cubic foot of water at 400 degrees, and weighing, at this temperature, 53.63 pounds. If the temperature of the range is to be kept at 350 degrees the water can only be allowed to fall to 350 degrees; that is to say, the permissible fall in temperature is 50 degrees, and the whole quantity of heat liberated by the cooling of the cubic foot of water is 53.63 times 50, multiplied (if we would be very exact) by 1.0174, the specific heat of water at this temperature, or 2728 heat-units.

The medium which is most commonly used, instead of water, for the transmission of heat, is steam. Supposing, instead of admitting water to the range, we introduce a cubic foot of steam at the same temperature of 400 degrees. If this steam falls from 400 to 350 degrees a portion of the steam is condensed, and the latent heat liberated. A cubic foot of steam at 400 degrees weighs 0.547 pound;

at 350 degrees it weighs 0.3056 pound; the difference between the two, 0.241 pound, is the weight of the steam condensed to water. The latent heat of evaporation of steam at 400 degrees is about 830 units per pound; therefore 830 times 0.241, or 200 heat-units, is the quantity of heat liberated by the condensation. To this might be added the heat given out by the fall in temperature of the 0.3056 pound of steam; but since the specific heat of steam at this temperature is only 0.31 this addition amounts to only 0.09 heat-units. We may say, then, that the cubic foot of steam at 400 degrees yields 200 heat units in cooling to 350 degrees; while the cubic foot of water delivers, under the same conditions, 2728 heat-units. Hence it is obvious that 13.64 cubic feet of steam must be circulated to do the same amount of heating as may be accomplished by one cubic foot of water. Just as soon as it has fallen to the temperature at which the range must be maintained the steam must be exhausted to give place to a new supply. It is true that steam, being a light aeriform fluid, will flow through pipes much more easily than water will. By the well-known laws of hydraulics the relative velocities at which fluids travel through pipes vary inversely as the square root of the densities. The relative velocity of water to steam, at the temperature of 400 degrees, is as $\sqrt{0.547}$ to $\sqrt{53.63}$ or 1 : 9.87. Consequently, under the same conditions, with the same length of pipe, the same resistance in the pipe, and the same pressure on the circulating medium, 9.87 cubic feet of steam would flow to one cubic foot of water. But the water is to the steam, as far as heat-carrying power for equal volumes is concerned, as 13.64 to 1. Hence, to deliver the same amount of heat under the conditions named, the current of steam should flow 13.57 as fast as the water-current; and since, with the same power propelling, it will flow only 9.87 times as fast, that power must be increased to secure the required velocity, which will be $13.64 \div 9.87 = 1.38$ times as great as the steam-velocity at the assumed power.

It must be remembered that transmission through a pipe is a question not of weight, but of volume. A four-inch pipe will carry no more *cubic feet* of mercury at a given velocity than it will of hydrogen gas, although the density of the mercury is several thousand times that of the hydrogen. So to deliver equal quantities of heat there must be in the case of steam a velocity of 1.38 times that due to its smaller density, as compared with water under the same conditions. The mechanical work, which is the measure of the expense of transportation of a fluid, varies as the cube of the velocities at which the

fluid flows. The relative expense of transporting equal quantities of heat, by steam or water under the conditions assumed, will, therefore, be as $(1.38)^3 : 1$, or the cost for steam will be 2.63 times that for water.

It is usually assumed that a current of steam flowing through a pipe is maintained by the expansive force of the steam itself. Precisely; but this expansive force in the steam is only attained by a fall in pressure and temperature, and consequently by a corresponding amount of condensation. Returning to our former example, if, at the end of a long line of pipe, it is wished to deliver steam at a temperature of 400 degrees, corresponding to a pressure of 250 pounds to the square inch, it would be necessary at the central station to put upon the boilers a sufficient pressure, in addition to that at which it is expected to deliver steam, to overcome the inevitable friction of the pipe between the boilers and the place where the steam is to be received. In a long line this friction is of considerable amount, so that, in order to accomplish the necessary delivery of steam, the boilers would be called upon to bear a burden equal to the amount of radiation of the line plus the amount of frictional resistance offered to the steam-current. The frictional resistance may, of course, be reduced to a minimum by the use, in the line, of pipes of very large diameter. This has frequently been done, with the inevitable result of very largely enhancing the cost of the plant, and increasing the difficulties both of construction and maintenance.

In the case of a water-plant it is only necessary to subject the boilers to a pressure requisite to give the temperature at which it is wished to deliver water, plus the much smaller amount of radiation which takes place from a pipe of less diameter than that employed in a steam-plant, the frictional resistance of the pipe being entirely overcome by means of the forced circulation obtained by the pump. The boilers, which perhaps are the most critical part of the system, being entirely relieved from this extra pressure, are much more easily constructed and maintained. Thus, by means of an incompressible fluid like water, and the employment of a pump to produce circulation, a much higher initial pressure can be placed upon the pipe-line to overcome the frictional resistances of the pipe, thus enabling us to employ a much smaller pipe than it is customary to use in steam plants, and largely decreasing the expenses and difficulties of construction and maintenance.

BIOGRAPHICAL NOTICE OF BYRON W. CHEEVER.

BY WILLIAM H. PETTEE, ANN ARBOR, MICH.

(Birmingham Meeting, May, 1888.)

A FEW weeks ago the Institute lost, by death, from its list of members one who in recent years, by frequent attendance at meetings, by the preparation of papers, and in many other ways, had shown a warm interest in Institute affairs; who had an extensive acquaintance among his fellow-members, and whose memory it is fitting to perpetuate by a permanent record in the volumes of our *Transactions*.

Dr. Byron William Cheever, acting Professor of Metallurgy in the University of Michigan, died at his home in Ann Arbor, on the 6th of March, 1888. He was born September 17th, 1841, in Ellisburg, Jefferson County, New York. In 1844, the family moved to Geneva, Illinois, and thence, in 1857, to Dundee, Michigan. Two years later, a third removal was made to Ann Arbor to take advantage of the educational facilities there offered; and in 1859 the subject of this sketch, with his two brothers, entered the University of Michigan. In 1863, he took the degree of Bachelor of Arts. After a year or two of business life, he entered the medical department of the University, and graduated as Doctor of Medicine in 1867.

A taste for scientific work had been developed in him in early life; and, even while carrying on undergraduate and medical studies, his proficiency in chemistry found recognition in his occasional employment as assistant in the University laboratory. The interval between the close of his course in Arts and the commencement of his course in Medicine was also devoted to chemical work, partly in the city of Philadelphia, and partly in the West Indies, where he was employed in the service of a commercial company engaged in the guano trade.

Dr. Cheever never took up regular practice as a physician. After receiving his medical diploma he went again to Philadelphia, and for about two years was employed as an analytical and consulting chemist in that city. A favorable opportunity occurring for work in the Rocky Mountain mining region, he left Philadelphia and

went to Georgetown, Colorado, where he had his home until 1872. In his three or four years of life among the mines, professional duties carried him to many of the more prominent mining districts of Colorado, and as far south as New Mexico.

As a young man, Dr. Cheever was considered to be unusually strong and rugged, and to possess a constitution that might be expected to stand up well under the exposure incident to mountain life. But the event proved otherwise; and in 1872 he returned to Ann Arbor, somewhat broken in health, and suffering especially from a tendency to rheumatism and to malarial affections, aggravated by the action on his system of the vapors of the mercury used in the metallurgical operations he had been conducting. Though not often kept from regular duty by acute disease, he was obliged for several years to be carefully watchful of his health, and to avoid all unnecessary exposure. Members of the Institute who have been familiar with his appearance at the meetings held in the last few years, will easily recall the evidences, obvious in his features and his general bearing, of a lack of full bodily strength and vigor.

After his return from Colorado, and while it seemed questionable whether it would be wise for him, in his impaired state of health, to attempt to do much work that would require his presence in a chemical laboratory, Dr. Cheever turned his attention for a time to the study of law. In 1875 he took a third degree, that of Bachelor of Laws, from his *alma mater* of 1863 and 1867. But law was not so much to his liking as science; and in 1878, having in the meantime, in fact, devoted a large share of his energy to chemical work, he accepted a position in charge of the instruction in quantitative analysis in the chemical laboratory of the University of Michigan, a laboratory in which he was already well known. The position was one that gave him occupation in congenial work, and at the same time afforded opportunity for further pursuing private studies and investigations. In 1881, his duties were enlarged, and he was appointed to the chair of metallurgy, which he filled in an eminently satisfactory and successful manner down to the time of his death. He was elected a member of the Institute at the Washington meeting, in February, 1882.

As a teacher, Professor Cheever readily and rapidly gained the confidence and esteem of his students. He neither required nor expected impossibilities of them; but he easily enforced the necessity of doing an ample amount of rigidly accurate work. He was possessed to an unusual degree of a native skill in the handling and

devising of apparatus for carrying out any experiment or research he might wish to make; and this facility of manipulation, coupled with the fact that he seemed always to have some practical aim in view, bore fruit in impressing the pupils under his charge with a sense of certainty that the university work they were asked to do by him could not fail to be of immediate benefit to them when called to take up problems in actual practice in the mines or mills of the country. It is a source of pleasure to be able to record, in this connection, that the hopes thus excited in the minds of his students were not often disappointed. Scattered as they have been, and are, from the east to the west, from Lake Superior to Tennessee and Alabama, his pupils bear constant and wide-spread testimony to the efficiency of the instruction they received under Professor Cheever's guidance.

Professor Cheever was never a voluminous contributor to professional literature, though in the last few years of his life he gave an ever-increasing amount of time to recording the results of original work, and, either for himself alone or in conjunction with others, prepared numerous short articles for publication in chemical and pharmaceutical journals, in addition to those that have appeared in the *Transactions* of the Institute.

Possessing, as he did, a wide familiarity with the conditions that affect the present and prospective value of mineral deposits, and the elements essential to success in industrial undertakings, and being endowed, moreover, by nature with strong common sense, a sound judgment, and an integrity of character beyond all question, it is not strange that his services as an expert were frequently required by persons seeking profitable investments in mining or metallurgical enterprises. Among his more recent professional engagements of this kind may be mentioned an examination he made in Honduras, of fields supposed to be rich in gold, and the ill-fated visit in February last to western Arizona.

From this last-mentioned expedition he hastened home, suffering from what appeared, at first, to be simply a severe cold; but within less than a week after returning to his family he was prostrated with acute typhoid malaria and sank rapidly away.

The Institute has lost an esteemed member; the university with which the greater part of his scientific career has been closely connected, and the community at large, in which he was best known, are called to mourn the loss, in the prime of life, of a teacher and a citizen whose valuable services in the past gave promise of still greater worth in the future.

METHODS OF MINING IN THE MENOMINEE RANGE, MICHIGAN.

BY JOHN FULTON, JOHNSTOWN, PA.

(Duluth Meeting, July, 1887.)

THE following table shows the shipments from the four Lake Superior iron-ore districts during the year 1887, and their total outputs from their beginning until the close of 1887.*

Name of District.	Output of 1887.	Total Output to close of 1887.
Marquette	2,033,102	25,562,102 G. tons.
Menominee	1,189,907	7,611,998 "
Gogebic	1,237,704	2,070,879 "
Vermilion	394,252	986.256 "
Grand total	4,854,965	36,231,235 G. tons.

It will be noted that the Menominee district is second in importance in its total output tonnage to the close of the year 1887.

These ore deposits are found in jasper-slates and in the horizon between these and their associated clay-slates; in the latter they are true contact-deposits. They are found in this jasper- and clay-slate formation under varied conditions, yet, in all, certain characteristic laws are preserved. The angle of dip varies from 15° to 20°, and up to 85° and even 90°, with the horizon. The ore is rather soft, bluish-black in color, and is a hematite. It varies in thickness from a few inches to one hundred feet, or more. It is mainly "Bessemer" ore, containing 57 to 63 per cent. of metallic iron, and .03 to .07 per cent. of phosphorus.

In the mining nomenclature of this range two conditions of the ore-lens are described by the terms "*dip*" and "*pitch*." The former is in common use in defining the relation of the ore-plane to the horizon; the latter indicates the posture of the ore-body as it inclines east or west from its position at the opening of the mine. In the

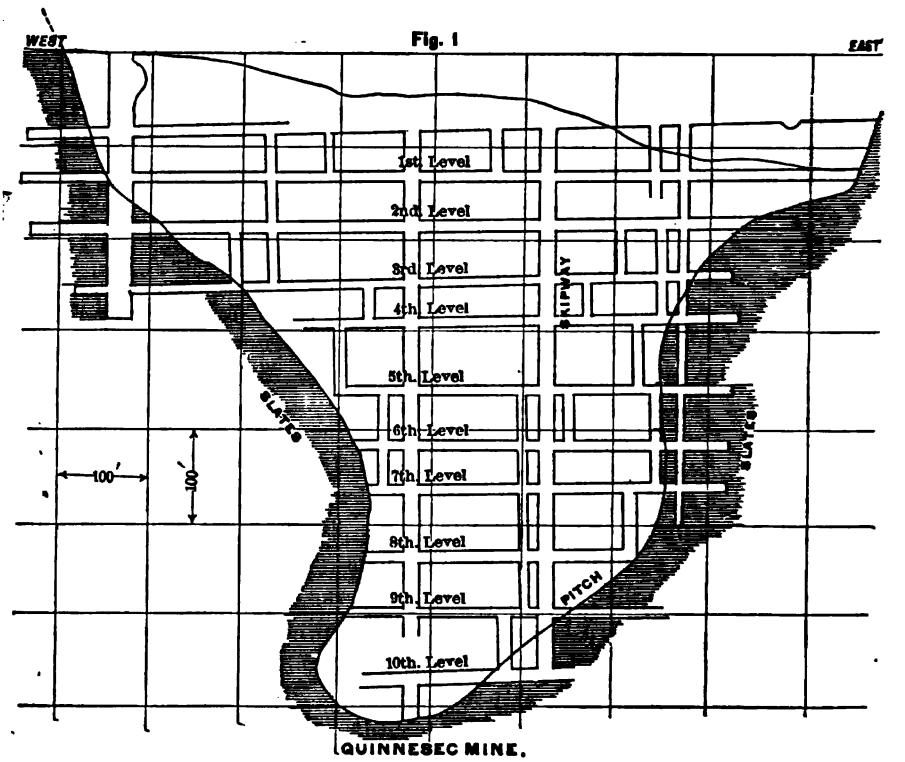
* "Statistics of Output of Mines." By Charles D. Lawton, Esq., A.M., C.E., Commissioner of Mineral Statistics, Michigan.

Menominee Range the "pitch" is usually westward, and varies from 27° to 45°.

It will thus be seen that the "pitch" of the ore-body is next in importance to the dip in enabling the mining engineer to locate the works intelligently.

Some of the ore-deposits outcrop to the surface boldly. Other deposits are found with the upper edges beneath the surface. Other

Fig. 1



conditions being equal, the latter can be relied upon for the greater depth, as the original volumes have not been reduced by the eroding or denuding agencies which cut down the former to varying depths, and consequently reduced volumes. But the conditions indicating large or deep deposits vary so much that they are not yet well understood.

Along the ore-bearing group of jasper and clay-slates the ore-deposits are found in three or more horizons, the Norway siliceous limestone formation affording the miner a base from which he can estimate the probable place of the ore in these measures.

Naturally, the deposits outcropping at the surface invited the first mining operations.

It might reasonably be inferred that, with the accumulated mining knowledge of the past, an advanced system would have been introduced at the very opening of operations on the Menominee Range; but such was not the case. The first mining methods were very crude. In the experience of ten years they have been rapidly advanced, until now they are fully abreast of the best modern mining practice. In all this upward movement in mining methods it is evident that necessity in a great degree compelled progress.

As human progress has been illustrated by the stone age, the bronze age and the iron age, so the growth of mining in this range has had its distinct periods of open pit-work, irregular timbering, Nevada system of framed timber, and the recent system of rock-filling.

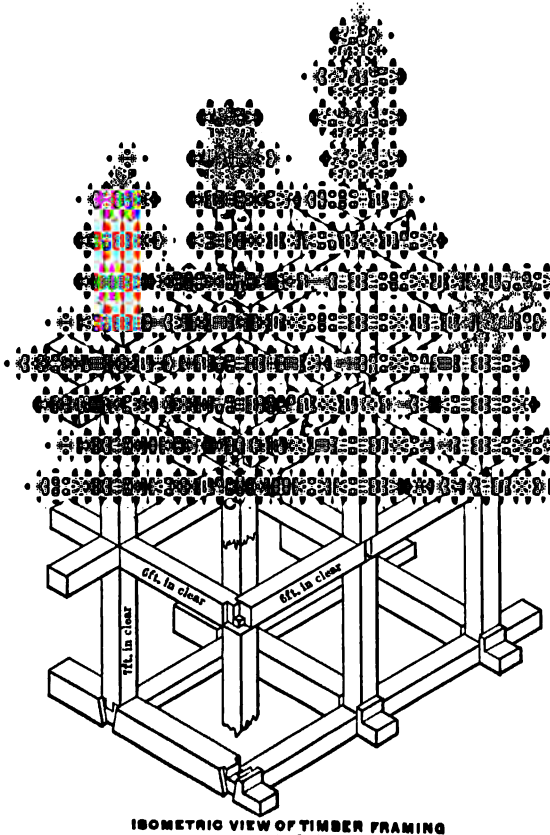
The open-pit mining embraced ore-deposits outcropping at or near the surface. These were mined with skips or derricks in a very crude way, so as to afford the utmost output, without reference to the economy of subsequent working or the stability of the mine.

This unplanned mining was also the result, in part, of the want of experience in the depth and posture of the ore-bodies.

There were justifiable instances in this open-pit mining, such as in the west end of the large Norway Mine, where an outcrop of ore over 100 feet thick and 150 feet deep was mined to great advantage. The Cyclops pits have been mined in the open way, near the surface, with horses and carts, and in the deeper portions, with the usual derrick and tubs, banking over 100,000 tons of excellent ore. But these are exceptional cases. The open mining work at other places proved wasteful and dangerous in every way.

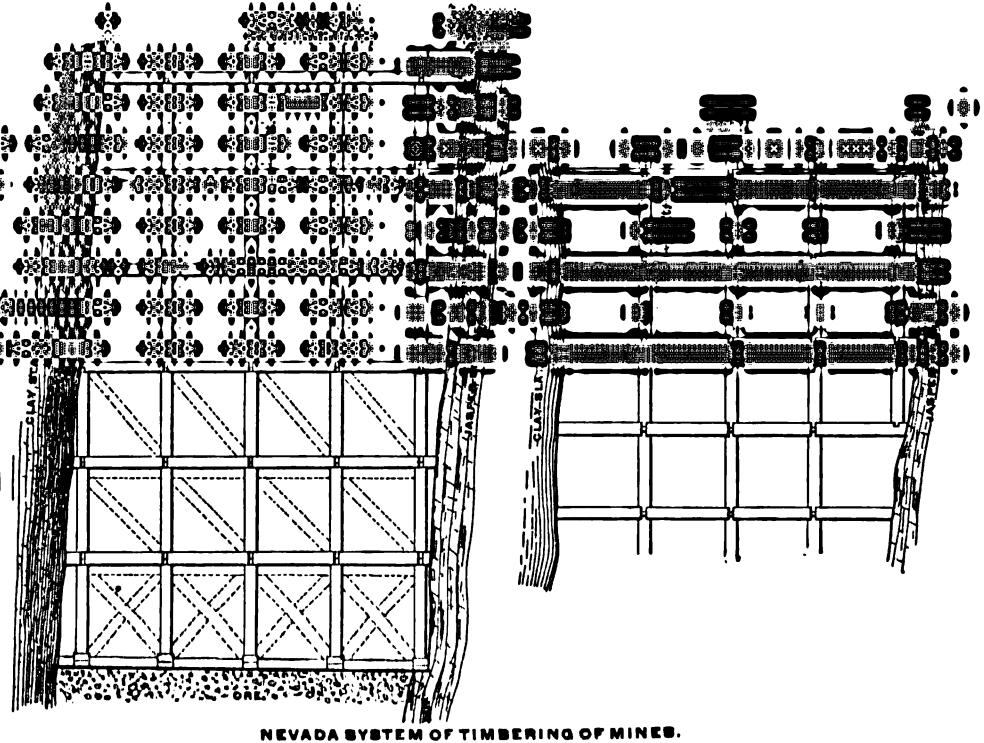
As the mining deepened, regular skip-ways were established and protected by timbering. At intervals down these skip-ways of 60 to 100 feet, levels were driven on each side and at right angles to the skip-ways. These are known as the *levels* of the mine, and are numbered 1, 2, 3, 4, etc., in descending order. This second period evolved some attempts at securing the hanging-walls by round timbers, placed across the exhausted space of the ore-body and at right angles to its bedding-plane. This early effort at timbering was also supplemented by leaving ore-pillars at certain places to steady the hanging-walls. Cribbing was also used in places where a crush or creep made its appearance.

The third period, induced by increasing depth and its accompany-



ISOMETRIC VIEW OF TIMBER FRAMING

Scale of Feet: 0 1 2 3 4 5 6 7 8



NEVADA SYSTEM OF TIMBERING OF MINES.

Scale, $\frac{1}{16}$ inch = 1 foot.

ing pressure, compelled, in most mines, the application of some *system* of timbering. The time at which the improved plan of timbering was introduced at the several mines along the range varied from many different causes.

In the large Norway mine the hanging-wall is of jasper-slates, tough and firm, and can be easily sustained with little support. In fact, this mine is made up mostly of a series of great caverns of ore, each affording 10,000 to 20,000 tons, with heights of stopes from 20 to 60 feet. Posts and crib-timbering were used in the early years of this mine. Mines in the ore-bodies, with jasper-slate foot-walls and clay-slate hanging-walls, were the earliest to require careful timbering and the first to compel systematic working.

Captain John Curnow introduced into the East Vulcan Mines of the Penn Iron Mining Company* the plan of timbering known as the "Nevada system," which consists in filling the space exhausted in the ore-body with a series of frame cubes. These cubes are constructed with white pine timber of excellent quality, 12 to 15 inches square, framed into squares of 7 feet from center to center. The drawings (Fig. 2) will show the value of this system of timbering as related to strength, facility in erecting, and its adaptability to all thicknesses of ore-deposits or variations in hanging and foot-walls.

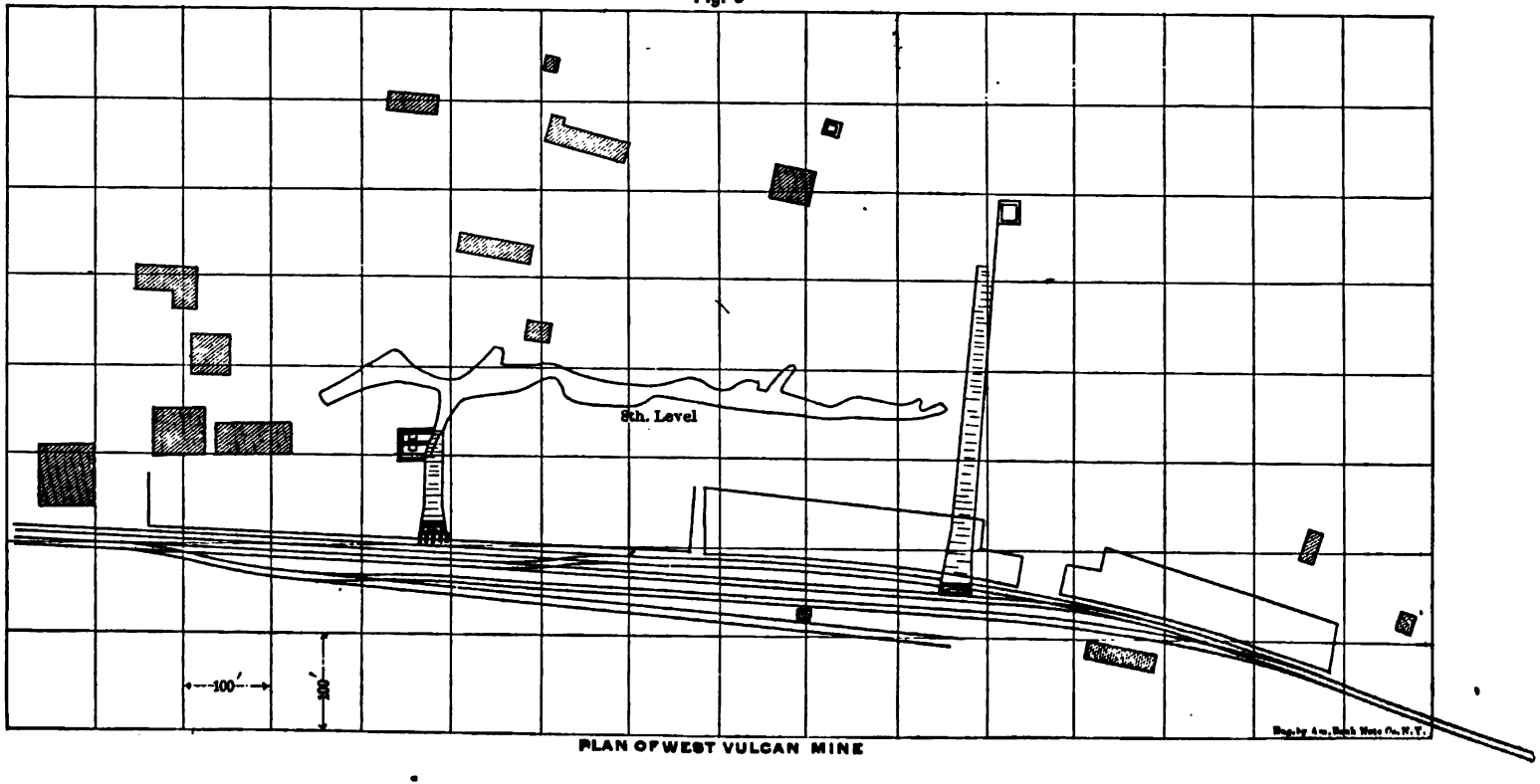
When the walls of the ore-deposit are tolerably firm and not easily softened by exposure to the moist atmosphere of the mine, this system possesses great strength; but where the hanging-wall is of soft clay-slates, as in West Vulcan, softening rapidly on exposure to the air in the mine, this system with its large timbers affords only temporary support. When the crush begins, the upright posts lose their vertical position, and many of the timbers are crushed into splinters. Collars and cross-braces are sometimes used to arrest a squeeze; but a brief respite only is secured in this way.

The flexure of a soft hanging-wall indicates the early crushing of timbers in the portion of the level exhausted of the iron-ore. It becomes then a struggle on the part of the miner to remove as much as possible of the ore out of the creeping level before the final crush comes to close out all mining operations.

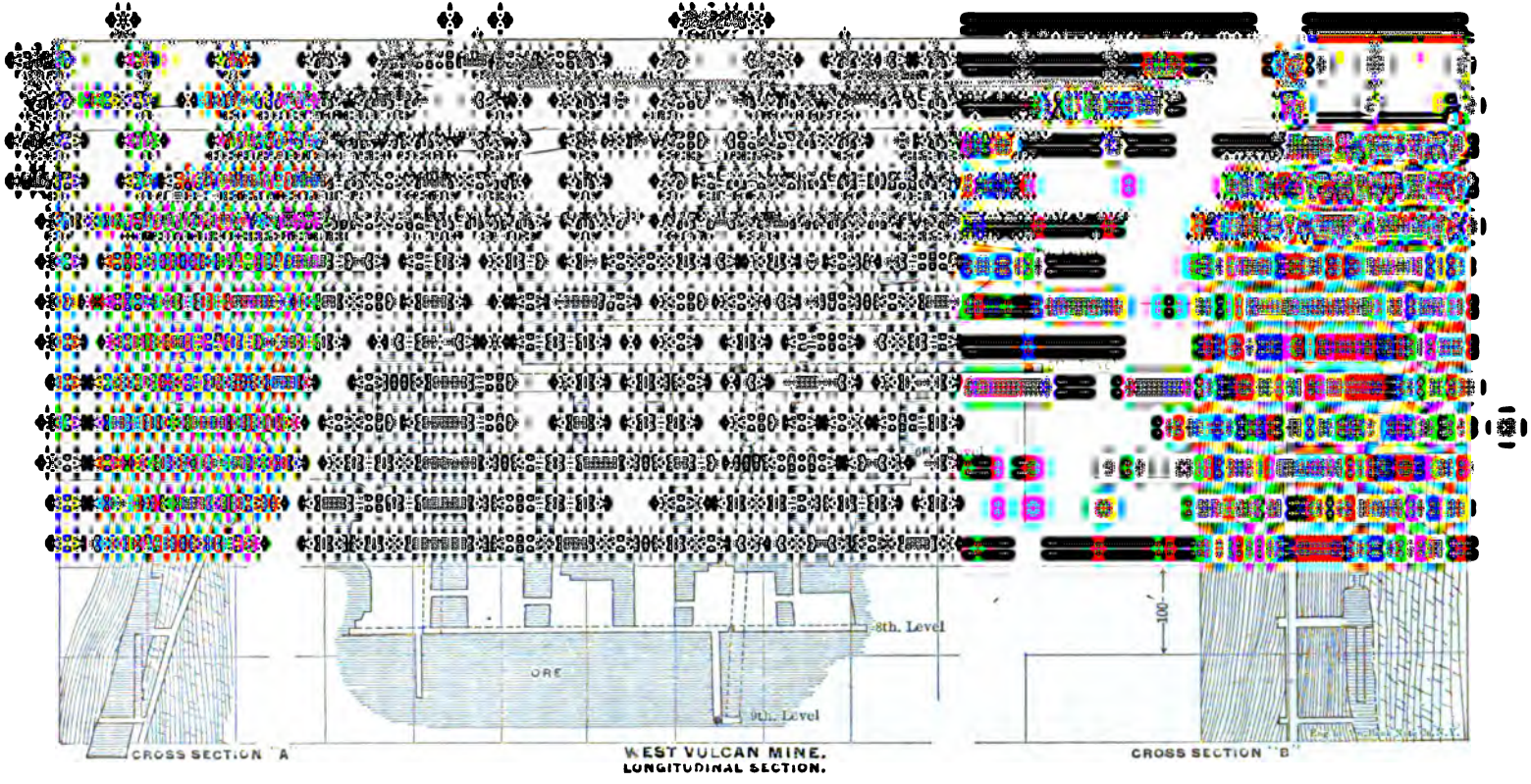
It has thus been found that for certain qualities of mine-walls this elaborate system of timbering fulfils its office satisfactorily, but that under a softening hanging- or foot-wall it is not reliable for a

* The Penn Iron Mining Company is the Michigan section of the Cambria Iron Company of Pennsylvania.

Fig. 3



PLAN OF WEST VULCAN MINE



WEST VULCAN MINE.
LONGITUDINAL SECTION.

sufficient time to afford opportunity for exhaustive mining in each level, or to protect ways to the adjoining level.

The depth to which this system of timbering can be carried safely depends on the firmness of the hanging- and foot-walls more than on the increased pressure from depth, although the latter is also an important factor.

It may be noted that with this timbering it became practicable to make exhaustive mining in each level, securing only the skip-ways by pillars of iron ore. Even this ore-support can be avoided when stopes or shafts are used.

The latest and modern method of "*rock-filling*" was compelled by the conditions already described, as in the West Vulcan mine of the Penn Iron Mining Company.

The drawings of this mine (Figs. 3 and 4) exhibit the general posture and westward pitch of the ore-deposit. At the eighth level the ore is 600 feet long and the average thickness about 25 feet.

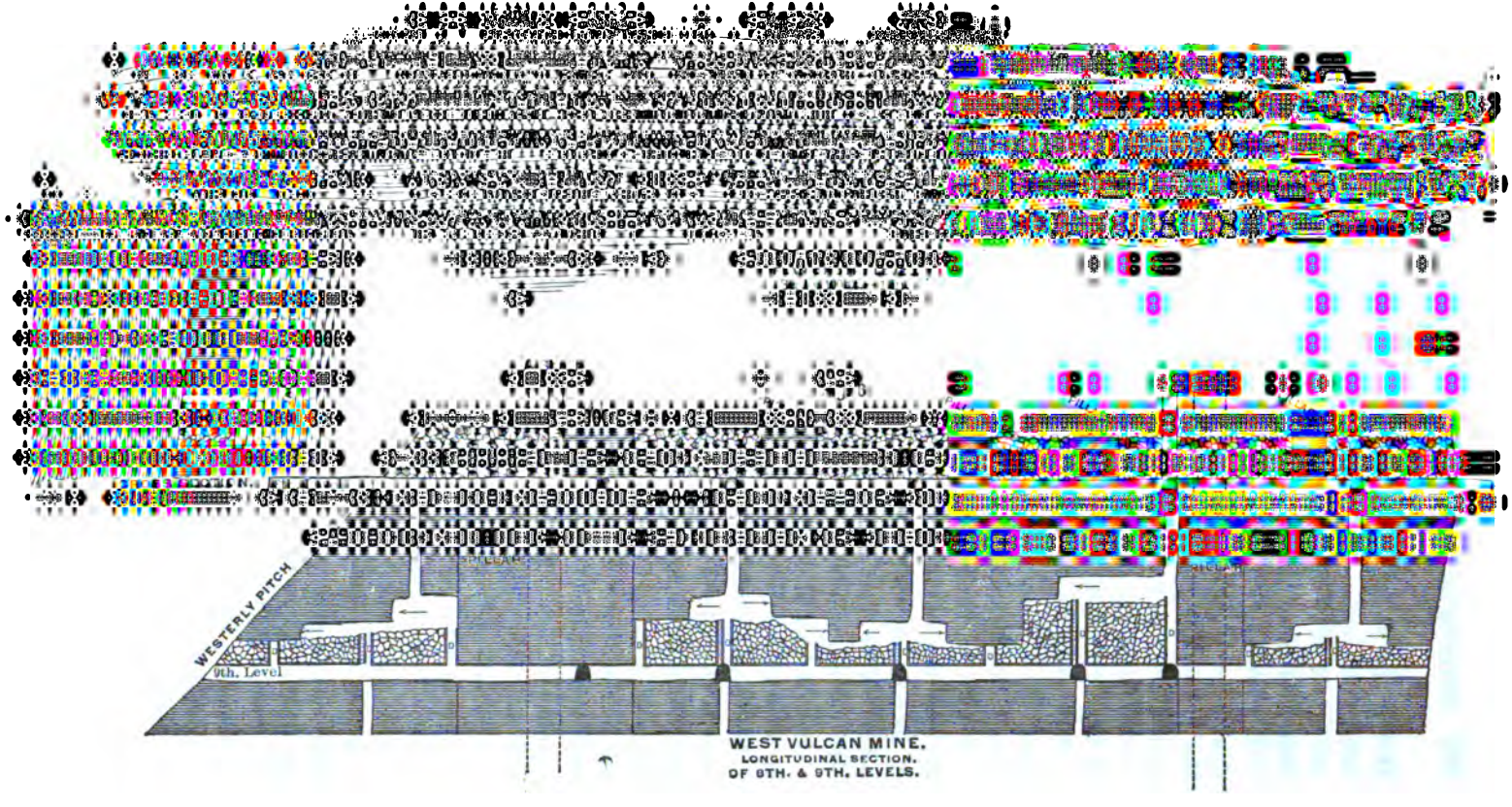
The longitudinal section exhibits the "levels," with the method of working the stopes and removing the pillars. This mine is operated by a shaft and a skipway or slope, as shown on the cross-sections. The large new shaft is 665 feet deep to the ninth level. The western pitch of the ore is clearly shown on this section.

The cross-sections looking west show the dip of the ore-body down to the ninth level, now the working-level. Cross-section B shows the main shaft in the hanging-wall.

The timbering, mainly after the Nevada system, has been carried to the eighth level. Rock-filling is now being used in the ninth level, from which 130,000 tons of ore, it is estimated, will be mined this year.

The drawings, Figs. 5, 6 and 7, show the method of this filling. From the main shaft "A," a drift cuts the ore-body and 25 feet into the foot-wall of firm jasper-slates. From this point a rock-tunnel is driven in the foot-wall east, as shown on plan, to shaft "B." Along this rock-tunnel ports are made at intervals of about 100 feet into the ore-body. From these ports the mining of the ore begins on the bottom of the ninth level.

The first cut of this ore is mined about 8 to 10 feet high and the spaces from which the ore has been removed are filled with rock. This rock-filling follows the mining and affords absolute safety and inflexible support to the walls of the mine. The broken rock for filling is conveyed down the winzes B, and the ore through the shutes D, which are built up as fast as the filling is made upwards.



WEST VULCAN MINE.
LONGITUDINAL SECTION.
OF 8TH. & 9TH. LEVELS.

It will thus be seen that the tunnel in the foot-wall secures absolute safety to the main-ways of each level; it is out of the crush range in any event. The winzes or rock-mills are also ventilators and greatly improve the sanitary condition of the mine.

The ore-body is, under this system, attacked by the miners on double face at each portal C, from the main rock-level or tunnel in the foot-wall. The mining by sections upwards is simply a repetition of the first 8 or 10 feet; the rock-filling following the mining work and filling the spaces from which the ore has been removed as rapidly as room for the mining operations will permit.

Some timbering will be required occasionally in this rock-filling, especially in sustaining the filling in upper levels, as it is approached from below in removing the last cut of ore.

It may be noted here, that this rock-filling system has not been evolved from any special progress in the mining art, or in the line of its economy over the Nevada system of timbering, but has been compelled by the absolute necessity of the case in West Vulcan and all other mines in this range where it has been introduced.

It may be said that sufficient work has not been done to submit an accurate statement of the relative expenses of these two systems.

However gratifying this comparison would be, if it indicated economy on the side of the rock-filling, yet this is not the vital factor governing the change of method from timber to rock, but rather the latter is a compelled requirement to secure the stability of the hanging-wall of the mine. And further, it may be pointed out that such a statement at any time would not afford a permanent basis of comparison, from the fact that the magnificent pine forests of this section of Michigan are fast disappearing under the great drain for these mines and eastern markets, so that the timber-supply is becoming year by year enhanced in value by scarcity and lengthened haulage. It is, therefore, a constantly shifting factor of expense in mining operations.

The rock, on the other hand, is inexhaustible; and, in spite of some little variableness in the cost of breaking it up and sliding it down into the mine, is a much less fluctuating factor of expense than the wood for timbering. The cost of timbers and timbering, per ton of ore mined at West Vulcan during 1887, was 38 cents.

The cost of rock-filling, with its necessary use of some timbering, per ton of ore mined, is estimated at 20 cents per ton—14 cents for rock-filling, and 6 cents per ton for the timbering required with

rock-filling. These estimates require verification by a few years of actual work.

It may be pointed out here, that the rock-filling affords a permanent support, which is not liable to decay as is the timbering system, and will not require renewals during the progress of the mine-workings.

The great caverns of ore in the Norway mine compelled rock-filling about two years ago, so that Captain John Oliver, of Norway, was one of the pioneers in the application of this system.

The Chapin was also forced to use this rock-filling system to maintain the walls of its great ore-deposit, and was the first on this range to apply it.

Many of the mines, with firm walls and at moderate depths, continue to use the timbers in steadying their works, but the rock system must commend itself rapidly to all these mines from its great safety, stability, and facility for exhaustive mining.

There are other important factors in connection with the general application of this system of rock-filling. The location of the shafts or slopes to the mines is one. The first planning of these slopes or skip-ways placed them in the ore-body, requiring large pillars of ore for their maintenance. Where the ore is of moderate thickness (10 to 15 feet), this method is not so open to criticism; but when the ore-body is 20 to 50 or 100 feet thick and rather soft, it has serious drawbacks in the great pillars of ore which must be set apart for this purpose, and in their tendency to crumble, especially in the event of a creep or crush, which follows in a greater or less degree in exhaustive mining.

The use of skip-ways in the foot-walls, or the sinking of shafts in the hanging-walls, are matters of the greatest importance in assuring safe and economical mining. It has been shown that slopes or skip-ways in the ore-body are open to serious objections in large deposits of ore.

It is quite possible to sink the slopes or skip-ways *in the foot-wall*, especially when this is composed of firm jasper-slates. They could be driven in the foot-wall, and a sufficient distance under the plane of the ore, say 10 to 20 feet, so as to assure a permanent rock tunnel under all conditions of creep or crush. But this system has some exceptions, as in the case of the East Vulcan mine, with its ore-body flexed and pitching, as shown in the longitudinal section, Fig. 8. It will be evident that a slope or skip-way in the foot-wall slates of an ore-body so placed would have been impracticable. But

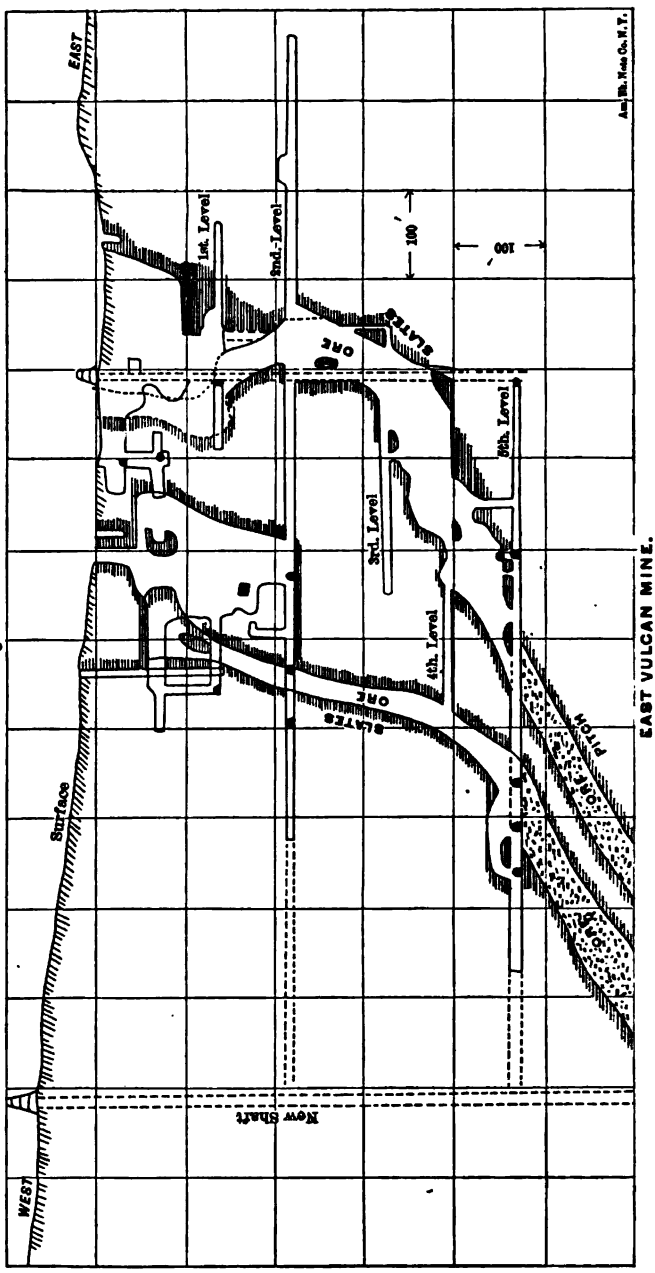


Fig. 8

EAST VULCAN MINE.
LONGITUDINAL SECTION.

there are many deposits where a slope or skip-way could be driven in the foot-wall of an ore-deposit, and be readily made to conform to the flexures met, in ordinary cases, in these foot-walls.

This method would be costly at first, but it would ultimately be found to assure great safety, and permit exhaustive mining, since no pillars would be required for its protection.

This East Vulcan mine, after passing through the early stages of mining, adopted the vertical shaft for winding and pumping. It is now 500 feet deep. At 400 feet deep, the ore-body pitched rapidly westward, requiring a long rock-tunnel from the shaft to the ore-body. It will readily appear that with the continued westward pitch, the lengthening rock-tunnels at each level could not be maintained, as their increasing cost would debar their use. Hence, in this case, after testing the pitch of the ore-body, it was concluded that a new shaft, 800 feet west of the old one, should be put down. Work on the new shaft is now in rapid progress. It is designed to produce the tunnels in the levels of the old workings, until they reach the new shaft, to assure good ventilation, and also to explore the intervening mining ground.

This new shaft will pass through the ore-bodies, and reach a point at which its levels or rock-tunnels will become too expensive if the present westward plunge of the ore continues. But it is also probable that the ore-body will change its present rather flat pitch. In any case, it is evident that sinking this new shaft in soft slates will be much less expensive than continuing the old shaft with its increasing lengths of rock-tunnels in jasper-slates.

This case has been selected as affording an example that has furnished an abundant series of perplexing mining problems to its managers.

West Vulcan mine exhibits (Fig. 4) an example of approaches by slope or skip-ways, partly in foot-wall slates, and a vertical shaft in the soft slates of the hanging-wall. The cross-sections of this mine will show the dip of the ore-body, and the relations of the shafts to it.

The location of the main shaft A in the hanging-wall was the result of several conditions bearing on the ultimate economy of mining its ore. It is near the siding of the railroad, affording a ready way for delivering the ore into railroad cars, and also for receiving coal by the same way for the boilers, and timbers for the mine. This location also reduced largely the height of the pumping column, and the shaft was sunk in soft slates rapidly and cheaply. Aside from these special conditions, the locating of a shaft in the hanging-wall

cannot be commended, as there is generally some shifting in the hanging-wall ground, endangering shaft and machinery.

It thus appears that a slope or skip-way in the foot-walls, with levels also in the rocks, is, in most instances, the safest and most economical plan.

It is evident, also, that the limited amount of exploring performed at most ore-deposits, fails to afford an adequate knowledge of the posture and pitch of the ore, and hence no comprehensive plan of mining operations can be laid down; the result is a shifting, undecided, and, in many cases, badly located system of ways, that must result in abnormal expenditures in operating them.

It has also been a matter of discussion whether the usual method of mining by beginning operations at or near the surface and working downwards, is the best plan. It is true that it affords a ready output of ore, and a quick return of money; but, unless permanent rock-ways are established, it involves increasing expenditure in the downward workings. It is submitted, that in deposits of ore of moderate thickness, a slope could be cut in the ore to the bottom of the deposit, and workings commenced there, entirely exhausting the ore in the progress of the working *upwards*. The exhausted spaces below would afford a ready place for slates or other mining refuse, and could, if necessary, be supplemented with additional rock-filling. Even should the hanging-wall swell or buckle, no serious injury could result, as such a crush would be arrested at each level.

The methods of shaft- and slope-timbering on this range are fully up to the best practice. In tunneling, compressed air and power-drills are used, giving rapid progress in extending the mine-ways. The winding and pumping plants at Norway and West Vulcan mines are models of comprehensive designs for this special work, in magnitude, efficiency, stability, and harmony of their parts. The shafts at these two large mines are 10 by 16 feet in size, and 400 and 700 feet deep, respectively. The double-acting plunger-pumps are 16 in. in diameter and 10 ft. stroke into single water column, throwing, when necessary, a continual discharge of water. The pumping-engines are 38 in. by 28 in. each, and the ponderous gear-wheels for driving the pumps are 30 ft. 8 in. in diameter. The winding-machinery, boilers, etc., are on a proportional scale.

Norway mine will probably produce 100,000 tons of ore this year, 1888, and West Vulcan will probably yield 130,000 tons of ore.

Improved systems of mining and modern plants of winding- and

pumping-machinery are becoming rapidly introduced in this range, especially as greater depths in mining are reached.

The western energy and push are admirably illustrated in the progress of improvement of mining methods in the Menominee Range. It is also hopeful to observe that the comfort of the miners is not neglected, and that at Norway and West Vulcan mines comfortable dry-houses for the use of the miners have been established, where not only the wet mining-suits are dried, but bath-rooms are provided also.

Reading-rooms have also been established at these mines. At Norway a hospital is open for the reception of miners for treatment under a skilled physician and surgeon.

The occasional fears of exhaustion of the ore-deposits of this range that marked its early experience, are now measurably abated under the compensatory elements of occasional discoveries of new deposits which sustain its annual output with assuring regularity.

The writer is indebted to F. A. Jansen, M.E., and chemist of the Penn Iron Mining Company, for the original drawings to illustrate this paper.

AN ALUMINUM-ORE.

BY EDWARD NICHOLS, DUNKIRK, N. Y.

(Duluth Meeting, July, 1887.)

As far as known to the writer, no discovery has heretofore been made in the United States of deposits of beauxite, or the hydrated oxide of alumina. The deposit which forms the subject of this note was discovered in Floyd County, Georgia. Numerous "float" specimens, covering an area of about half an acre, indicate the location of the main deposit, which has thus far been opened to an inconsiderable extent only. The excavations show the beauxite to exist apparently as large masses in clay. The formation is Lower Silurian, as determined by the Geological Survey of Georgia. The surface in the immediate vicinity is covered with numerous angular fragments of chert, which is a characteristic rock throughout this formation in Georgia. An examination of the mineral shows it to have the oölitic structure common to several beauxites. It varies in color from a light salmon to dark red, depending upon its contents of iron oxide. The light-colored specimens are comparatively soft, while the dark mineral is much harder, spots in the latter being

harder than quartz. Its chemical composition, as shown by the analysis given below, is interesting by reason of the presence of titanic acid, in which it resembles the same mineral as found in Asia Minor. It dissolves with difficulty in acids, but fuses easily in bisulphate of potash.

Owing to the comparative purity of this deposit it seems likely to have a value, before long, for use in some of the aluminum reduction processes and as a refractory material.

ANALYSES.

	I.	II.	III.
SiO_2,	2.80	.....	2.300
Al_2O_3,	52.211	57.248	56.883
Fe_2O_3,	13.504	3.212	1.490
TiO_2,	3.520	3.600	3.551
H_2O,	27.721	.....	.....
CaO,	none	.....	.....
MgO,	none	.....	.....
P_2O_5,	.....	.....	.066

Specific gravity, 2.35.

I. Dark specimen. II. and III. Light-colored specimens.

PETROLEUM AND NATURAL GAS IN NEW YORK STATE.

BY CHARLES A. ASHBURNER, PITTSBURGH, PA.

(Duluth Meeting, July, 1887.)

INTRODUCTION.

THE occurrence of oil- and gas-springs in the State of New York has been a fact of historical record since 1627, when the existence of the Cuba oil-spring was first recorded. The utilization of natural gas at Fredonia in 1821 attracted the attention of the public to the possibility of obtaining an illuminating gas by digging and drilling into the rocks. No active search was made, however, for oil or gas by the drilling of wells until after the discovery of commercial oil in Pennsylvania in 1859.

In 1862 a well was drilled at Bradford, McKean County, Pennsylvania, but a few miles south of the New York state line, to the depth of 200 feet, in search of the oil-sand which had been found at Titusville, Oil City, and at other points in Western Pennsylvania. Immediately subsequent to this, other wells were drilled in search

of petroleum at a number of points in New York, particularly in the western part of the State. Some explorations have been made, in an unsystematic way, for oil ever since, explorers being much encouraged in their search by the development of the Allegany oil-district in 1879 and 1880. Immediately subsequent to the general utilization of natural gas in Pennsylvania in 1882 and 1883, a new interest was taken in the drilling of exploration wells, more particularly in search of gas.

The principal object of this paper is to publish a record of the explorations which have been made up to date for oil and gas in the State. I have not mentioned every well which has been drilled, but only the more important ones with which I am familiar. The record of these wells have a two-fold value, first, to the practical geologist and well-driller, to enable him to deduce conclusions as to the possibility of getting natural gas in special localities, and to aid in further explorations; and second, to the technical geologist, in giving definite facts having a direct bearing upon the stratigraphy of the Palæozoic rocks throughout the State.

In communicating these facts to the Institute, it is not my purpose to state the conclusions at which I have arrived, as to the possibility of getting commercial oil or gas anywhere within the State, since my professional relations with active explorations in different portions of the State prevent me from making public any of the practical conclusions at which I may have arrived. Nor is it my intention to review at length the bearing which these facts have upon the geological structure of the State, for two reasons: first, because I am at present collecting additional facts which will have a direct bearing upon such an investigation, and which might modify any suggestion as to geological structure which I might now make; and, second, because my professional engagements will not permit me to devote sufficient time to the question. Again, to publish a complete report on such a geological investigation would extend the present communication beyond the limits of a suitable paper for the *Transactions* of the Institute.

To one familiar with the large and unwarranted expenditure of money in the drilling of wells for oil and gas in the State, it is surprising that the State Geological Survey, which has done so much good work in the advancement of palæontological and geological science, has never made a practical report on the stratigraphy of the State, or published geological county maps, both of which would have proved of inestimable value to oil-, gas-, and other mineral

explorers, and prevented a useless expenditure of money by the citizens of the State sufficient to pay for the cost of such a survey a thousand times over.

The history of natural-gas explorations in the State is naturally divided into two periods: first, that up to the discovery of the Allegheny oil-field in 1880; and, second, that subsequent to this date. Although the historical references to the existence of oil and gas, published prior to 1881, have no direct bearing in aiding present explorations, yet the records are so scattered, and have been so contorted from time to time by numerous publications in the local papers that it becomes important in this place to refer to them in detail.

Early References to Oil and Gas.

The first record of the existence of an oil-spring in the United States was by a French missionary, Joseph de la Roche-d'Allion, under date of July 18th, 1627, when he made mention in his diary of visiting the Cuba oil-spring in Allegany County. The location of the spring was noted on a map published fifty years later, being designated by the words, *Fontaine de bitume.* Vanuxem, in his Report of 1837 refers to this spring, and Dr. Beck, in his Report on the Mineralogy of New York, published in 1842, describes the spring as follows: "It is a circular pool, about eighteen feet in diameter, filled nearly to its margin with foul water, and having no visible outlet. The water is coated with a thin layer of mineral oil, giving it a yellowish-brown color, but the quantity is inconsiderable. The peculiar odor, however, is perceptible for a considerable distance from the spring. The substance is collected by skimming it from the fountain and is used as an external application in various diseases. Indeed, so highly was it prized by the Indians that a mile square around the spring has been reserved for the Senecas."

The following references to the occurrence of natural-gas springs in different parts of the State are taken from Dr. Beck's report:

"A locality occurs in the town of Northeast, in the County of Dutchess where, from the bottom of a small lake, proceeds inflammable gas of considerable purity. A short distance from Ameniaville in this County, the same gas rises from the bed of a small stream.

"In New Lebanon, County of Columbia, one or two similar springs have been observed.

"Inflammable gas is quite abundantly evolved through a boring made into the slate in the southern part of the city of Albany. The

gas which issues from the Albany spring burns with a white flame tinged with red; and at length, when the flow of gas is temporarily impeded, the flame is of a blue color, owing probably to the imperfect combustion.

"Carburetted hydrogen gas issues through a crag or gravelly soil about a mile west of the village of Vernon, in Oneida County. According to Prof. Eaton, who made several trials in July, 1823, it issues through a spring of water at the rate of a gallon in a minute. He states that he observed it 'issuing from several small masses of water along the foot of the same hill; which naturally induced a belief that it rises from the earth in all parts of several acres of ground adjoining the chief spring.'

"Gas is evolved in considerable abundance in various parts of Ontario County. The most noted locality is in the town of Bristol, about nine miles west-southwest from Canandaigua, where it issues in a ravine on the west side of the valley of Mud creek.

"Dr. Hayes, from experiments which he made upon this gas, arrived at the conclusion that it consists principally of the light and heavy carburetted hydrogen; and that it contains carbonic acid but no sulphuretted hydrogen.

"At Cheshire, five miles east of the preceding, there are several places from which jets of this gas issue from clefts in the rock. It is observed, however, in much the largest quantities, according to Mr. Hall, in Manchester, on the east side of Canandaigua lake, its occurrence being manifested for a considerable distance along the same range.

"In Yates County, a mile or a mile and a half from Rushville, in the town of Middlesex, Dr. Hayes, in the paper already quoted, states that on the southeastern side of a valley called Federal Hollow, there are numerous jets of this gas. 'In a field near the northern extremity of this tract, and at an elevation of forty or fifty feet from the bottom of the valley, several hillocks may be seen of a few inches in height, and from two or three to ten or twelve feet in diameter. They consist of a black mould and are destitute of vegetation; from these gas issues, and on digging into the earth beneath it may be obtained in considerable streams. These hillocks have appeared successively, within a period of seven or eight years, although the ground has been cleared much longer. The first indication of their formation is the disappearance of vegetation at a particular point; this is enlarged, the mould accumulates, and the hillock is gradually formed. In very cold weather the gas is said

to issue from the tops of hollow cylinders of ice, sometimes two or three feet in height, forming when lighted in a still evening a beautiful illumination.'

"In Monroe County, in the town of Riga, inflammable gas rises from a spring in sufficient quantities to supply a constant flame from a half-inch tube.

"In the town of Royalton, Niagara County, six and a half miles east of Lockport, carburetted hydrogen issues through the water of a basin on the south side of the Erie canal. This is said by Prof. Eaton not to have been observed until the water was let into the canal. Upon testing this gas, it was found to resemble that at Vernon, but it approached nearer to the character of light carburetted hydrogen. The quantity which issued through the basin at one place exceeded a gallon a minute. This place was named Gasport.

"At Van Buren Harbor, on Lake Erie, four miles from Fredonia, bubbles of inflammable gas may be seen rising through the water when the lake is calm, a rod or two from the shore.

"In the town of Sheridan, six and a half miles from Fredonia, the same gas is abundantly evolved in various places. And again, on the west branch of Canadawa creek, four miles southeast of that village, the carburetted hydrogen rises in considerable quantity through a spring of pure water in a marsh. This gas, when collected and forced through a small orifice, burns with a bluish-white flame.

"In Cattaraugus County, Mr. Hall states that carburetted hydrogen gas escapes from almost all the waters, whether stagnant or otherwise. It is evolved in small quantities at the oil-spring at Freedom, but it will not sustain a constant flame.

"To this account it may be added that carburetted hydrogen rises through several brine-springs, as at Clyde in Wayne County; in the valley of Elk creek, three and a half miles from the village of Delhi, Delaware County, and at La Grange, in Steuben County."

General Geological Considerations.

All of the natural gas which has been found in the State comes from some of the porous strata belonging to the Palæozoic system, with the exception of a small amount of gas which is contained in the porous sand-beds of the glacial drift. The detailed facts which are given in this communication, for each locality where any considerable amount of gas has been obtained, or where systematic ex-

Chart showing the Divisions and Prominent Thicknesses of the Rocks of the Palæozoic System in New York, [illegible]
by Charles A. Ashburner.

plorations have been made for it, show from what strata of the Palæozoic system the gas is derived. In order, however, to give a comprehensive view of the succession and thickness of the rocks of the Palæozoic system throughout the gas-producing areas of Ohio, Pennsylvania, and New York, I have grouped on the accompanying chart a number of general sections of these strata. The thicknesses given for the different formations show wide differences, and in themselves are sufficient proof of the necessity of geological knowledge in any drilling operation, either in search for oil or gas, or in the practical development of districts where either oil or gas or both are known to exist.

In all explorations it is important to know in what geological formation drilling is commenced, and how far above the lower limit of the formation the well is started. It is no less important to know the thicknesses of the different groups of rocks through which the drill is passing, in order to make an estimate as to the proper depth to drill to reach any definite horizon in which the existence of oil or gas can be expected. From an inspection of the accompanying chart it will be seen that the absolute thickness of any one series of strata has not been uniformly determined throughout the series of sections. This is due to the fact that it is impossible in some localities to determine the exact position of both the top and bottom of special formations. There are, however, in this section, certain geological horizons whose positions have been determined in each section, for instance, the thickness of the strata included between the bottom of the Catskill sandstone, No. IX., and the top of the Upper Helderberg limestone at the bottom of No. VIII. extends uniformly throughout the entire series. In the Catskill region in New York, the thickness of this series is 4420 feet; in the eastern part of the anthracite coal-fields in Pennsylvania it is only 3140 feet; along the Susquehanna River in Clinton County, Pennsylvania, 2440 feet, and in Huntingdon County, Pennsylvania, the strata attain their maximum thickness (in the three States for which the sections are given) of 5135 feet. In central New York they are 3912 feet thick; in western New York, 3500 feet, and in southern Crawford County, Pennsylvania, 3000 feet. In the upper part of these series of rocks occur the oil- and gas-sands of northern Pennsylvania and southwestern New York. A still wider variation in the range of thickness is shown in the Catskill sandstone, No. IX. This formation is interesting on account of its being geologically equivalent to the Venango-Butler group of rocks which contain all

the oil- and gas-sands in western Pennsylvania, around Pittsburgh; in this region the Marysville gas-sand forms the top of the group and the Butler fourth oil-sand forms the bottom of the group. In the vicinity of Pittsburgh the vertical thickness of this group is 500 feet. In the Bradford oil-district, Pennsylvania, and in the Allegany oil-district, New York, it is only 250 feet, while in Clinton County, Pennsylvania, it is 2100 feet and in Huntingdon County, 2680 feet; along the Lehigh River, in Luzerne County, the group attains its maximum thickness of 7145 feet. In the Catskill Mountain region, New York, from which this group takes its name, the thickness is only 2900 feet. Another notable variation in the thickness of the same rocks exhibited by the accompanying chart, is in the thickness of the Devonian and Silurian strata in central Pennsylvania and central New York. In central New York these strata are 9000 feet, and in central Pennsylvania 22,000 feet thick.

Geological Map of New York.

The accompanying geological map of southern and central New York and northern Pennsylvania, I have compiled from the maps of the Geological Survey of Pennsylvania, from my own field-explorations* in the State of New York, and from scattered records of Professor James Hall and other New York geologists. See Plate I.

The map is especially interesting to oil- and gas-explorers, when studied in conjunction with the sections given on the accompanying chart, and the detailed sections given in the description of local explorations. The most interesting fact, which is best brought out by this map, is that the Catskill Red sandstone, No. IX., which represents the Venango oil and gas-sand group of Pennsylvania, outcrops in northern Pennsylvania and southern New York, and that it is therefore impossible to hope to get oil or gas in northern Pennsylvania and southern New York in the same geological horizon from which the oil and gas in the vicinity of Pittsburgh is obtained.

Conditions Attending the Occurrence of Natural Gas.

Although it is impracticable, as I have stated, for me at the present time to publish the conclusions at which I have arrived as to the possibility of gas occurring in commercial quantity in different sections of New York, yet a brief reference to the general conditions

* The most extended explorations which I have made in New York have been in the Mohawk valley, in the Catskill Mountain region, and from thence southwest to Pennsylvania, and in the southern part of western New York.

controlling the existence of natural gas will be of special interest, in conjunction with the detailed facts contained in this paper.

At the time of the formation of the sand and limestone beds in which natural gas is now found, there were undoubtedly buried some of the organic remains from which both oil and gas were subsequently formed; but I believe that a large part of both the oil and gas has come from the organic remains buried in the associated strata, and that the porous beds in which they are now found merely act as reservoirs to hold them. There is no question but that the largest deposits of natural gas will be found in regions in the vicinity of which petroleum is found. Although the geological and physical conditions under which natural gas has been obtained in commercial quantities vary to a considerable extent in different localities, yet the most important conditions attending its occurrence may be enumerated as follows:

First. The original extent of vegetable and animal life at the time of the deposition of the sedimentary strata in which these remains were buried or of which they were formed.

Second. The existence, in close proximity to the strata containing such organic remains, of a porous and more or less homogeneous sandstone or shale, or a porous and cavernous limestone included in the rocks. Such porous strata serve as reservoirs to hold the gas resulting from the decomposition of the organic remains referred to. In many cases, the rocks in which a large amount of the organic remains were buried serve also as a reservoir to hold the gas.

Third. Solid strata, undisturbed by cracks and crevices, overlying the gas-reservoir rocks to prevent the escape of the confined gas into the atmosphere. In cases where the gas contained in the reservoir-rock has come from the organic remains buried below the reservoir, it is necessary that there should be cracks in the underlying rocks in order to permit the gas to flow into the reservoir.

Fourth. A moderate dip of the strata (the most favorable dip for the occurrence of gas being at the rate per mile of less than 75 feet), and the position of the anticlinals or saddles, and synclinals or basins in the strata associated with the gas-reservoir rocks.

Fifth. The relative proportion of water, oil, and gas contained in the gas-reservoir rock.

Sixth. The pressure under which the gas exists before being tapped by wells; this pressure is now generally called the rock-pressure, to distinguish it from the flow-pressure, or the pressure under which the gas flows from the wells.

A popular impression exists that the only rocks in our geological scale which contain carbonaceous remains are those of the Carboniferous age. This impression is probably due largely to the fact that this name has been assigned to the formation which contains the largest coal-deposits in Great Britain, in the central and eastern parts of the United States, and elsewhere.

In the geological ages prior to the Carboniferous, there did not exist sufficient land-vegetation to form extensive coal-beds; but the fossil remains of water-plants amply attest the fact that there was buried in the rocks below the Carboniferous a great abundance of vegetable forms. Even in the Laurentian rocks of Canada, far below where I believe it is possible to find natural gas, there is a large accumulation of carbonaceous material in the form of graphite, which is now universally conceded to have been derived from the remains of vegetation. These plants belonged to the lower forms of vegetable life, as the animal remains of which many of our limestones are composed belonged to the lower forms of animal life. The latter, no doubt, are the source of the large amount of both oil and gas derived from the limestone beds. According to researches made by Professor Peckham, in southern California, the petroleum and gas there are probably derived from microscopic animals.

No less important to the existence of gas or oil is the occurrence of a porous rock, to act as a reservoir, to hold the hydrocarbon products formed by the distillation of the organic remains. In the case of the oil and gas of Pennsylvania, a porous sandstone or sandy shale always acts as the reservoir-rock. This is also the case in the Allegany district, in New York, and in eastern Ohio. In western Ohio, Indiana, and Kentucky, the reservoir-rock is invariably a porous and cavernous limestone. Professor Orton claims that the porosity of the gas-producing limestone in Ohio "is due to the interlocking of its crystalline growths." This may, in a small degree, account for the porosity of the rock; but, from my examination of the pieces of the gas-rock brought up from producing gas-wells at Findlay, and of the limestone from other wells, and also of the Trenton limestone along its outcrops in Pennsylvania and New York, I believe that the porosity of the limestone is due almost entirely to small cavities which have been excavated in the rock by percolating waters. These cavities, in some places, are extremely minute.

The relative capacity of the pores of the Trenton limestone to the bulk of the rock itself has never yet been fairly determined for any oil or gas-producing district deriving its product from a lime-

stone bed. Mr. Carll has estimated that a good productive oil- and gas-sand in the Venango group can hold one-tenth of its bulk of oil: assuming that the oil-sand was only 15 feet thick in the most productive portion, it would contain, on an average, 9,600,000 barrels of oil per square mile. From my own estimates, I have determined that the oil-producing sand in the Venango district, up to the present time, has only produced an average of 900,000 barrels of oil per square mile. It is impossible to determine the absolute thickness of the reservoir-sand over any definite area, even as small as that of one square mile. There are so many variable factors which enter into the problem that it is difficult to arrive at any fair conclusion on the question of relative porosity. From estimates which I have made of the producing capacity of the oil- and gas-sands of the different Pennsylvania and New York districts, I believe that the amount of oil which will be ultimately retained, by capillary attraction, in the pores of the sand-beds, will range from eight to nine times as much as the total amount of oil which it will be possible to extract from these sand-beds through wells.

The existence of a solid impervious rock-roof over the gas-reservoir rock is necessary to the retention of the gas in the rock. Experience so far goes to prove that very few gas-wells have been obtained with a minimum daily capacity of a million cubic feet of gas in territories where the wells are less than 400 feet deep; whereas, some of the largest gas-wells have got gas from reservoir-rocks with only 800 to 1000 feet of covering.

That the absence of both petroleum and natural gas in plicated strata in Pennsylvania, east of the Alleghany Mountain crest, is to be explained by the cracking of the rocks, is evident, since the survey of the outcropping rocks, and a study of the records of dry wells, show that the oil- and gas-sands extend far beyond the limits of the area of the region in which any traces of oil or gas have ever been found. Even within the area where oil- and gas-wells have been found, the cracking or jointing of the rocks must have a potent influence upon the amount of oil or gas obtained in certain localities. From surveys which I have made in Elk County, Pennsylvania, it appears that the direction of certain streams is to be attributed to their flow along joints in the rocks, which have resulted from the contraction of the earth's crust during the process of secular cooling. I believe that this, in a measure, accounts for the occurrence of gas at certain points in western Elk County, and its absence at other points, the gas being obtained where the rocks are not jointed, and not being found where they are jointed.

The tendency of natural gas, petroleum, and water—which are invariably associated, to some extent, in all oil- and gas districts—to separate according to their respective specific gravities, makes it essential that the dip of the reservoir-rock and the position of anticlinals and synclinals shall be taken into consideration in locating oil- and gas-pools and oil- and gas-wells. If the reservoir-rock were uniformly porous and homogeneous over the arch of the anticlinal, the crown of the latter being level, and its cross-sections, transverse to its axis, being entirely similar throughout its extent, the best gas-producing territory would, of course, always be found on the crest of the anticlinal, immediately over the anticlinal axis, and the so-called *anticlinal theory* would always be a safe and unvarying guide for the location of a gas-pool. Such conditions are never universal in any oil or gas-district with which I am familiar: there exist in all gas-pools many modifying and purely local conditions which must enter into the consideration of the best localities in which to drill a well either for oil or gas. It is impossible, in this connection, to treat this subject at length, but I will mention two important modifying conditions: If an area underlaid by porous rock be of limited extent along an anticlinal axis, but of greater extent across an anticlinal and its adjoining synclinals, and if the pores of the rock be filled almost entirely by natural gas, the natural gas will, of course, be found in any well which may be drilled into the porous rock under this area, irrespective of the relative position of the well to the anticlinal axis or to the synclinals. Again, if the crest of the anticlinal, instead of being horizontal, dip along its axis, and the porous rock passes over the anticlinal and its adjoining synclinals on either side, a gas-well might be obtained in the center of either synclinal if the elevation of the porous rock in the center of the synclinal be higher than the elevation of a point in the porous rock on the crest of the anticlinal. For instance, if a gas-well be found on the crest of an anticlinal at any point, and the dip into the center of an adjoining synclinal one thousand feet from the anticlinal axis be twenty feet, and the dip along the crest of the anticlinal in a distance of five thousand feet be fifty feet, the chances for getting a gas-well at the point indicated in the synclinal are much greater than the chances of getting a gas-well on the crest of the anticlinal, five thousand feet distant from the producing well.

Where the rocks have a minimum dip they have necessarily been least disturbed since the time of their deposition, and in these localities we find the cover to the gas-sand less broken by cracks than

in other localities where the dips are greater and the synclinals and anticlinals more defined.

The relative proportion of water, oil, and gas, and the pressure under which they exist in the reservoir-rocks, must, of course, affect the amount of gas which can be obtained in any given time from a well, and also the duration of the well. This is too great a subject to be fully treated in this place; but it may be sufficient to state that the amount of gas which can be commanded per day from a given gas-territory, and the duration of the gas in the large gas-districts, such as that surrounding Pittsburgh, and in Warren County, Pa., and in the Findlay district, Ohio, will certainly be as great as has been claimed by Professors Lesley, Carll, Orton and myself, who have given this subject the most careful consideration. That the bulk of the gas which can be obtained for commercial purposes now exists and is stored up in inexhaustible reservoirs, I have no doubt; still, at the same time, there is a slow accretion to the supply constantly going on, which is a product of the continuing distillation of carbonaceous matter contained in the rocks. This product is so small, however, that it can hardly enter into the consideration of the question from a commercial standpoint.

Popular Plans of Explorations.

About a year ago extensive operations for the exploration of natural gas were planned by a gas-company, organized for the purpose, in northeastern Pennsylvania and southeastern New York. The plans and operations of this company fairly illustrate the unsystematic plan pursued by similar organizations in many localities in the United States, and are worthy of note. As far as I can gather, the plans of this company have been based upon information gathered from various sources.

In the first place, it was currently reported that a Pennsylvania geologist had succeeded several years ago in tracing the oil- and gas-sands of the Venango group in Western Pennsylvania east as far as Susquehanna, Wayne, and Pike counties in northeastern Pennsylvania. Whether this report was founded upon any statement made by a geologist or not, the fact is that although the geological horizon of the Venango oil- and gas-sands does exist under Susquehanna, Wayne, and Pike Counties, yet the sands themselves have thinned out and do not exist east of McKean County, which is 110 miles west of Susquehanna County.

In the second place, certain gas-springs were found to exist in the Delaware valley, principally in the vicinity of Port Jarvis, which

naturally emphasized the report in regard to the Venango oil- and gas-sands. That these springs are in themselves any indication of the existence of natural gas in commercial quantities, our experience does not bear out.

The results of the drilling operations of the company, as far as they have been reported to the public, are of interest as showing where natural gas does not exist in commercial quantities, but judging from the location of the wells and the character of the strata drilled through, they certainly throw no light on the question as to where possible gas-pools may exist in the vicinity of the wells which have already been drilled.

The fact that the managers of this company are reported to comprise experienced operators in the oil-fields and Philadelphia capitalists, and that they will drill other wells until their territories of 50,000 acres, more or less, shall have been thoroughly explored, fails to inspire confidence in the enterprise. I question their wisdom in adopting a policy, the motive of which seems to be to determine by costly drilling the territory which does not contain gas, rather than to determine by a judicious, systematic, and intelligent method whether any gas-pools exist under their properties which contain gas in commercial quantities and at points where it can be commercially utilized. If I am correctly informed of the location of this territory, a careful systematic drilling of two or three wells under the direction of an engineer and geologist experienced in the occurrence of natural gas, for the entire property, with a small expenditure of money, would no doubt settle the question of the existence of natural gas in commercial quantity; whereas it is currently reported that the company had expended, up to December, 1887, between $40,000 and $50,000 in drilling wells without having arrived at any practical conclusion as to whether it is *possible* for any gas to exist under its territory.

The Fredonia Gas-Wells.

The earliest published reference to natural gas in the State of New York was to the gas-springs near the village of Fredonia, on Canadaway Creek, in Chautauqua County, about three miles south of the Lake shore. Gas was first utilized here about 1821. Dr. Beck, in speaking of these springs in 1842, says: "The gas-springs seem to have their origin in the strata of slate which form the bed of the stream, and which are everywhere met with in this vicinity, a short distance from the surface of the earth." At one point where more than the usual discharge of gas was observed issuing from a fissure in the rock, a shaft was dug through the slate to a depth of twenty-

two feet. This shaft was closed and the gas was conveyed by a pipe from the shaft to a gasometer near by, which had a capacity of 220 cubic feet and could be filled in about fifteen hours with gas sufficient to supply from 70 to 80 lights. The gas was used for illumination in an adjoining mill, in several stores, and in a hotel. The history of the Fredonia gas-district is interesting because this is the first natural gas which was profitably used.

The following is a statement contained in the History of Chautauqua County, the accuracy of which is vouched for by persons who have been associated with the Fredonia Natural Gas Company:

"The use of natural gas in Fredonia was begun in 1821, when experiments were made to determine its illuminating value, and it was introduced into a few of the public places, among which was the hotel that then occupied the site of the Taylor House, and which was illuminated when Lafayette passed through the village in 1824. The gas so used at that time was the first used in the United States, and the gasworks established here were the first in this country. The spring first discovered, and from which the gas was first used, is located on the north bank of the Canadaway Creek, at the bridge crossing the stream at Main Street, in the village of Fredonia. The gas escaped at various places in the immediate vicinity, but when the well was sunk it was all drawn to it. The gas from this well, which was sufficient for thirty burners, was used alone until 1858, when another well was sunk on the creek in the northwest part of the village, by Preston Barmore, the shaft being thirty feet deep, six feet in diameter at the top and fourteen feet at the bottom, with two vertical borings, one of 100 and the other of 150 feet depth. In the fall of 1858, Elias Forbes purchased a half interest in the well, and that fall a company was formed, and during the remainder of that and the following year the gas, in sufficient quantity to supply about 2000 cubic feet per day, was conducted to the village through three miles of mains and supplied directly from the well to the stores of the village. During the latter year (1859) the company put in a gas receiver or holder of 12,000 cubic feet capacity, and supplied private houses. In the fall of 1871 Alvah Colburn made a boring for gas near his mill, with a view to supplying fuel for generating steam therefor, but the supply was inadequate for that purpose, though it was evolved in considerable quantity. He therefore purchased the Barmore interest in the gas company and connected his well, which is 1200 feet deep, with the company's receiver. Since that the supply of gas has been ample for the demands of the village. Previous to the opening of Colburn's well the supply of

gas was not sufficient to meet the demands for it during the winter, and the deficiency was made up by gas manufactured from coal. Professor Hadley's experiments show that the consumption of natural gas as compared with that manufactured from coal, through burners of equal capacity and in equal times, is less than one-half, with a greater candle power. He shows that a burner which consumed six feet of coal gas in one hour, with an illuminating power equal to fourteen sperm candles, six to the pound, consumed of the natural gas a fraction less than three feet, with an illuminating power of a little more than sixteen sperm candles. The natural gas also possesses a greater diffusive power, and one who has been accustomed to the use of coal gas finds it difficult to read ordinary print without being in close proximity to the light, is astonished at the facility with which he can read in any part of an ordinary-sized dwelling-room under the light from the natural gas."

The Fredonia natural gas is still used extensively for illumination, the candle-power of the gas being much greater than that of the natural gas found in the vicinity of Pittsburgh. Mr. E. J. Crissey, Secretary of the Fredonia Natural Gas (Light) Company, in a private letter in 1882, says: "The gas from this well (Colburn), which was sufficient for about thirty burners, was used alone until about 1858, when another well was drilled which supplied some two hundred burners. Another well was drilled in 1871 with better success. The average monthly supply of the three combined is about 110,000 cubic feet, of which an average of 80,000 cubic feet per month is consumed for lights. Seven other wells, varying from 50 to 800 feet deep, have been made without success. The area covered by these wells is about one mile in length by one-half mile in width. The supply has not perceptibly diminished since the opening of the wells."

The report of the Fredonia Natural Gas Company shows that in 1886, in a district containing a population of 3000, the price of gas per thousand cubic feet was $2.50; 72 public lamps were supplied with gas, at an annual price of $12 each. During that year no record was made of the total annual consumption of gas.

Mr. John F. Carll reports the Fredonia gas to come from strata immediately above the Corniferous limestone. The following are the facts reported to Mr. Carll by Mr. Alvah Colburn, President of the Gas Company:

"In digging the conductor to the rock some gas appeared. After the drill was introduced the gas increased all the way down to 700 feet, below which point no further increase was obtained.

"The limestone (Corniferous) was struck at 1050 feet and continued until the drill was stopped at 1207 feet. It was 'hard and flinty,' and produced neither oil nor gas. The well is tubed at 700 feet. The lower part of the hole fills up to the tubing with salt water. The pump is put in motion about once a year, but there seems to be no accumulation of water above the bottom of the tubing.

"The flow of gas is 4000 cubic feet per day by measurement. Pressure 19 pounds per square inch. The well was drilled in 1871 or 1872, and is apparently delivering as much gas now (October, 1877) as when first struck.

"Mr. Colburn kindly gave me some specimens of limestone, one of which, coming from a depth of 1200 feet, contains a well-preserved fossil shell, readily recognized as the Atrypa prisca, figured in *Geology of New York*, vol. iv, p. 175."

It is of interest to note here the character of the Corniferous limestone; and no better description can be found than that given by Professor James Hall in his report of 1843, as follows:

"The color of the Corniferous limestone varies from a light grayish-blue to a dark blue or black, and it is sometimes even of a light gray or drab color. It contains numerous nodules of hornstone, and the strata are sometimes separated by irregular layers of the same. In other localities these layers of hornstone increase in number and thickness to the almost entire exclusion of calcareous matter, and they then present a very harsh outline. At the eastern end of the district the hornstone is intermingled and interstratified with the calcareous strata, the whole very dark colored. The same character prevails at the western extremity of the district, where the rock outcropping on the Niagara has, from its black color, given name to the village of Black Rock."

The Corniferous limestone at Black Rock is 52 feet above the surface of Lake Erie, or 625 feet above tide. In the Colburn well at Fredonia, the Corniferous limestone was found in the well at a depth of 1050 feet or 315 feet below ocean-level, the top of the well being 735 feet above ocean-level, so that the average dip of the limestone from Black Rock to Fredonia, a distance of 38 miles in a direction south, 35 degrees west, is 25 feet per mile. According to Prof. H. S. Williams, at Gowanda, in northern Cattaraugus County, about 20 miles due east of Fredonia, the Corniferous limestone is not far from 1000 feet below the surface. The upper 700 feet of this interval is composed of Portage and Genesee shales, and the lower 300 feet of Hamilton and Marcellus shales. From a comparison of the outcrops in the northern part of New York and in Canada

West with the records of the Wyoming County salt-wells, Prof. Williams concludes that the Helderberg and Salina formations together range from 450 to 500 feet thick, or including the Corniferous limestone the thickness would be 600 feet. From this he concludes that the salt horizon of Wyoming County should be struck below Gowanda at 1600 feet. He further estimates, from his examination along the Niagara river, that the aggregate thickness of the Niagara, Clinton, Medina, Hudson River and Utica formations is not less than 1500 feet, making the Trenton limestone at least 3100 feet below the surface at Gowanda. From these facts the following would be an approximate section of western New York in the vicinity of Gowanda:

Portage and Genesee,	700 feet
Hamilton and Marcellus,	300 "
Corniferous,	100 "
Lower Helderberg and Salina,	500 "
Niagara, Clinton, Medina, Hudson River and Utica,	1500 "
Total,	3100 feet.

Composition of Fredonia Gas.

A number of analyses have been made of the Fredonia gas. In 1886, a French geologist, M. Foucou, collected a sample of the gas which was analyzed by M. Fonqué and reported to be a mixture of marsh gas (CH_4), ethyl hydride (C_2H_6), with a small quantity of carbonic acid and 1.55 per cent. of nitrogen. In 1887, Professor Francis C. Phillips made an analysis of the Fredonia gas for the Geological Survey of Pennsylvania, with the following results:

The sample of gas analyzed was taken from the mains of the Fredonia Natural Gas Light Company, May 12th, 1887.

Two determinations of nitrogen in this gas gave 9.58 per cent. and 9.50 per cent. respectively. Mean, 9.54 per cent.

In two determinations of carbon dioxide there were found 0.38 per cent. and 0.44 per cent. Mean, 0.41 per cent.

Results of Analysis of Fredonia Gas.

Nitrogen,	9.54 per cent.
Carbon dioxide,	0.41 " "
Olefines,	none.
Carbon monoxide,	"
Free hydrogen,	"
Ammonia,	"
Hydrocarbons of the paraffin series,	90.05 per cent.
	100.00

343.47 cubic centimeters of Fredonia gas yield on combustion:

H_2O — 0.6254 gm., corresponding to H — 0.06964 gm. = 21.83 per cent.
CO_2 — 0.9144 gm., corresponding to C — 0.24938 gm. = 78.17 " "

100.00

Making allowance for the 9.95 per cent. of nitrogen and carbon dioxide contained in the gas, it is calculated that the 90.05 per cent. paraffins present contain:

Per liter — 0.80627 gm. carbon.
0.22515 gm. hydrogen.

In a second combustion of Fredonia gas, 326.17 cubic centimeters yielded:

H_2O — 0.5927 gm., corresponding to H — 0.0660 = 21.89 per cent.
CO_2 — 0.8635 gm., corresponding to C — 0.2355 = 78.11 " "

100.00

As these quantities of carbon and hydrogen belong exclusively to the paraffins in the gas, it is calculated that the paraffins—amounting to 90.05 per cent. of the total gas—will contain:

Per liter — 0.80185 gm. carbon.
0.2247 gm. hydrogen.

In these calculations an allowance is made in the determination of the carbon for the very small quantity of carbon dioxide which always occurs in the original gas.

The means of the two results above cited are, per liter of paraffins:

0.80406 gm. carbon, ... = 78.14 per cent.
0.22492 gm. hydrogen, ... = 21.86 " "

100.00

In the above case, no tests were made at the wells. A test made at one of the wells in August, 1884, showed traces of oxygen. In the limited quantity at disposal for the above analysis, no positively certain indication for oxygen could be obtained.

Williams's Conclusions for Western New York.

Professor H. S. Williams, whose conclusions as to the section of these strata have been quoted above, says:

"The character of the rocks, as far as known, would indicate that more or less gas would be found for the first seven or eight hundred feet down. This is the place in the series for the Devonian black shales, and as far eastward and northward as Livingston and Genesee counties whenever they come to the surface the shales are strongly

permeated with petroleum odor. But above the black shales the first coarse sandstones in the above-named counties and in Wyoming County are always strongly scented with petroleum when freshly taken from the quarry.

"Also there are frequent gas-springs, dotted all over the southwestern part of the state, and occasionally oil-springs, the source of both the oil and gas of which is doubtless these black shales.

"In Western New York, although there are slight low folds, the whole inclination of the rocks is southward, and there is no reversal of the dip to form an anticlinal or large dome until the limits of the state are reached.

"From these facts it results that if there were gas originally formed in the Trenton, under Western New York, there is nothing in the geological structure to suggest that any considerable reservoir of gas is there at the present time. Such are the indications from the study of the surface-rocks of the region.

"In the southern part of Cattaraugus County, both from the higher elevation of the surface and by the gradual dipping under of the strata, the black shales, with the capping sandstones (Portage or other layers in the Chemung), are low enough to be reservoirs of oil and in some cases also yield gas."

The Buffalo Gas-Wells.

The Buffalo Cement Company has drilled, since 1883, four wells in order to test the presence of natural gas, salt-water or salt-rock or other valuable minerals under its property, consisting of about four hundred acres of land, situated wholly within the city limits. A number of valuable facts relating to these explorations have been communicated to me by Mr. L. J. Bennett, President of the company.

Well No. 1 was drilled in 1883 with a diamond drill. A complete core of the rocks passed through is now in the possession of the company, but will ultimately be deposited in the Museum of the Buffalo Society of Natural Sciences. This well showed gas in a very limited quantity at a depth of 451 feet, 9 inches, which slightly increased in volume down to 490 feet, 6 inches, where the drilling ceased. The gas-rock, as shown by the cores, is a very compact sandstone with numerous pin-point openings.

Well No. 2 was put down, in 1884, to a depth of 1305 feet, but proved unsatisfactory. No salt was found and but little salt water; and no perceptible increase of gas was obtained beyond that shown

in well No. 1. Mr. Bennett writes that from the best information based upon his drill-notes, the various rocks penetrated in well No. 2 were as follows:

Lower Helderberg limestone,	50 feet.
Salina shales,	550 "
Niagara limestone and shales, Clinton sandstone and shale,	185 "
Medina sandstone,	520 "
Total,	1305 feet.

This well and No. 3 were drilled with a 5§-inch jump-drill.

In May, 1887, a third well was drilled in which more gas was obtained than in either of the previous wells. The gas-rock, at a depth of about 460 feet, was shot with a 40-quart torpedo of nitro-glycerine, after which the amount of gas sensibly increased. The gas was shut in the well and a steam-gauge attached to the casing, which indicated a pressure of 60 pounds per square inch in 15 minutes, and 142 pounds per square inch as the maximum confined pressure. In December, 1887, six months after the completion of the well, a meter-test was made of the amount of gas discharged at the end of an inch and a quarter pipe, 865 feet from the well, and it was found to be 27,600 cubic feet per 24 hours. The gas is reported strongly impregnated with sulphur, but gives satisfactory results as a fuel. This well is 517 feet deep, having passed through 50 feet of Lower Helderberg limestone and 467 feet of Salina shales. Mr. Bennett reports that the company is now drilling well No. 4, and has under contract a fifth well, with the hope of obtaining more gas than in the first three wells. It is also proposed to drill No. 2 deeper, until the Trenton limestone is encountered. As far as I am aware, however, no reliable estimate has been made by an experienced geologist as to the depth to which this well would have to be drilled before reaching the Trenton limestone.

Dr. Julius Pohlman, Director of the Museum of the Buffalo Society of Natural Sciences, has made a careful study of the strata passed through in wells Nos. 1 and 2. Referring to well No. 2, he says:

"Beginning at a spot where the rocks of the water-lime group, suitable for the manufacture of cement, had been removed, and which is 70 feet above the level of Lake Erie, or 643 feet above tide-water, the drill encountered the following strata:

1–25 feet, shale and cement rock in thin streaks;
25–30 feet, tolerably pure cement rock;
30–43 feet, shale and cement rock in thin streaks;
43–47 feet, pure white gypsum;
47–49 feet, shale;
49–61 feet, white gypsum;
61–62 feet, shale;
62–66 feet, white gypsum;
66–73 feet, shale and gypsum, mottled;
73–131 feet, drab-colored shale with several layers of white gypsum, measuring 18 feet in all;
131–133 feet, dark-colored limestone;
133–137 feet, shale and limestone;
137–140 feet, dark-colored compact shale;
140–720 feet, gypsum and shale, mottled and in streaks;
720–725 feet, limestone;
725–760 feet, soft red shale;
760–785 feet, white solid quartzose sandstone, very hard;
785–1305 feet, soft red shale.

"At 1305 feet the drill was stopped. Permanent water was struck at 43 feet; gas of fair quality as well as quantity, at 452 feet; salt water, leaving on evaporation about 12 per cent. of salt, was found at 555 feet. A shaft, 20 feet square, was sunk on the premises later, for the purpose of determining the feasibility of mining the gypsum, but the rush of water through the gypsum layer at 43–47 feet, was so strong that a pump with a capacity of 2000 gallons per minute failed to make any impression upon it, and the attempt was abandoned.

"The average dip of the rocks from the north to the south, in this vicinity, is about 20 feet to the mile. The top of the Niagara limestone, ten miles north of Buffalo Plains, is about 20 feet above the level of Lake Erie, or 593 feet above tidewater; hence, as the drill started at 70 feet above the lake level, Niagara limestone ought to be found at a depth of about 250 feet; but in fact, the drill penetrated down to 1305 feet in the soft red shale, characteristic of the lower part of the Onondaga Salt Group, according to Hall, demonstrating that the Niagara limestone, if present, has between Niagara Falls and Buffalo, a dip of at least 130 feet to the mile."

Dr. Pohlman's opinion that the drill in this well did not penetrate below the Salina measures, and that the dip of the Niagara limestone between Niagara Falls and Buffalo is at the rate of 120 feet

to the mile, has been questioned by a number of local geologists. As I have made no personal investigation on the ground of these two questions, I am not prepared to express a professional opinion.

Oil and Gas in Allegany County.

No geological report has ever been published on the oil- and gas-sands of the Allegany district; in fact, no careful survey has ever been made of its geology. All the geological information which is available is that which has been gathered during the progress of private surveys made by Mr. John F. Carll and myself. I have made a number of examinations in the district during the past twelve years, and have sufficient data in my possession to arrive at definite conclusions as to the details of the geological structure. It is impossible in this place to give all of my facts or to make a complete descriptive geological report, but a few facts given here will be of special value for a clear general understanding of the geology, not only of the oil- and gas-district, but of Allegany County. Before giving these facts, I will state two conclusions at which I arrived over five years ago and which make clear the connection between the geology of this district and that of the Bradford oil-district in McKean County.

1. The geological horizon of the Allegany oil- and gas-sand which is commonly and locally known as the Richburgh is the same as that of the main producing oil- and gas-sands of the Bradford region, known by the oil-well drillers as the Bradford third sand, although it has no direct structural connection with the Oil Creek third sand, which is more generally known simply as the "Third Sand" and which occurs, geologically, 1000 feet above it. The Bradford oil-sand at the city of Bradford lies 1030 feet below sea-level; the elevation of the top of the sand above sea-level being 414 feet.

2. The oil-sand struck in the old Waugh and Porter well No. 1 on lot 34, Bolivar township, at a depth of 1330 feet, is undoubtedly the same as the Bradford oil-sand, and consequently the same as the Richburgh sand.

These two conclusions are not only of interest to the geologist but of great practical value to the oil- and gas-operator. The position which has been generally assigned to the Allegany or Richburgh oil- and gas-sand in the geological column is about 300 feet above the Bradford sand. This conclusion of the well-drillers and a number of local geologists has been based upon the fact that the lithological characteristics of the strata above the producing sand in

the Allegany district and in the Bradford district are different. The sandy measures in the 1000 or 1500 feet immediately overlying the oil-sand of Bradford and Allegany are poor guides in looking for oil and gas-sands in new or wild-cat territory around either the Allegany or Bradford oil- and gas-districts. They lead to confusion, error and disappointment. It is true that in a limited territory within the Bradford district there are distinct sands 300 and 600 feet respectively above the Bradford oil- and gas-sand, but I am quite confident that it is impossible to determine the position of the main producing oil- and gas-sand by the location of these upper sands. The correctness of my conclusions as to the geological relationship existing between the Bradford sand, the Allegany sand and the Waugh and Porter sand will, I think, be acknowledged after a careful study of the few facts given here.

The geographical location of the oil- and gas-wells referred to in Allegany County may be ascertained by reference to the accompanying map (Plate III).

On lot 29 in Genesee township, the top of the Richburgh oil- and gas-sand was struck in the old Cranston well No. 1 at a depth of 1704 feet; the elevation of the well being 2225 feet above tide. The same sand was struck in well No. 2 at a depth of 1709 feet; the elevation of this latter well above tide being 2235 feet. Less than a quarter of a mile southeast of well No. 1 occur detached blocks of sandstone and conglomerate from the outcrop of the Olean conglomerate. Some of these blocks range from 20 to 30 feet on a side and are 15 feet high. As nearly as could be ascertained from a careful study of these rocks, the elevation of the bottom of the Olean conglomerate, of which these rocks are the broken-down outcrop, is 2255 feet.

These facts show that the top of the oil-sand is 1729 feet below the bottom of the Olean conglomerate. The oil-sand in the Cranston wells is without doubt the same as the oil-sand which has proved so productive in the old Davis wells on lot 31, Genesee township; in the old Davis and Haldeman wells on lot 24, Genesee township, in the wells on lot 17, Clarksville township, and also in the Armour and White wells on lot 24, Genesee township. The elevation of the top of the oil-sand in the wells referred to, which is given in the accompanying table, proves conclusively that this sand is the same as the oil-sand generally productive throughout the entire Alleghany district. At Bradford the oil-sand is found 1779 feet below the bottom of the Olean conglomerate, so that the rocks composing

the interval between the oil-sand and the Olean conglomerate thin down only 50 feet toward the northeast in a distance of twenty-six miles between Bradford and Bolivar. This is less than might have been expected from an inspection of the thickening of the rocks composing this interval from Bradford south and southwest. The elevation of the oil-sand in the Waugh and Porter well is 500 feet above tide; the elevation of the bottom of the Olean conglomerate above this well is 2200 feet; so that the vertical distance between the Olean conglomerate and the Waugh and Porter sand is 1700 feet, as against 1729 feet in the western end of the Allegany district, and 1779 feet at Bradford, thus proving conclusively that the productive sands of these three localities occupy the same geological horizon.

The principal value of this conclusion can be appreciated when it is known that a hope has always been entertained by many of the well-drillers and operators in the Allegany field, of finding a second productive oil-and gas-sand in that field 300 feet below the Allegany gas-sand, and representing the Bradford oil-sand. Many thousand dollars have already been uselessly spent in drilling wells with this object. This conclusion, privately communicated for the first time to a number of operators in 1883, and published for the first time about a year ago, proves the folly of continuing this search further. It may be accepted as beyond question that the productive oil- and gas-sands in the vicinity of Richburgh, Bolivar, Allentown, the Waugh and Porter well and at Bradford are geologically the same, although they differ much in their physical characteristics.

The first well drilled in Allegany County of which I have any record was at Independence. This well was sunk in 1865 by a stock company, and a slight showing of oil and gas was obtained in a thin sandstone about 300 feet above what afterwards proved to be the Allegany oil-sand. The next well was drilled by Tadder & Company with similar results, a year or two later. In 1878, Clark & Company, of Scio, put down the third well near the second. This well was drowned out, and the fourth was drilled by Collins & Clark during the same year. All of these were located in the northwestern part of Independence township. In September, 1877, the Honeoye or Alma well, on lot No. 25, south Alma township, was commenced by the Wellsville & Alma Oil Company and finished in November. It was drilled to a depth of 1800 feet, cost $4000, and proved a failure. At a depth of 500 feet considerable gas was obtained, which ultimately took fire and burned down the derrick.

At a depth of 1000 feet, a small amount of oil was obtained, but a torpedo failed to increase the yield, and the well was finally abandoned.

The discovery of oil in commercial quantities in the Bradford district in 1874, and its active development, which commenced in the latter part of 1875, stimulated drilling in the surrounding region. Mr. O. P. Taylor, a native of Virginia, but at the time a resident of Wellsville, Allegany County, proposed to drill in the vicinity of the town in 1877. In 1878 he persuaded citizens of Wellsville to unite with him in drilling a well on lot 26, Alma township. The well failed to get oil in paying quantities, and succeeded only in proving the existence of an oil-sand. This was sufficient encouragement to the drilling, on lot 118 in the same township, of a second well, known as Pikeville No. 1, which was completed in November, 1878. The elevation of the top of this well is 1775 feet, and the top of the oil-sand was struck at a depth of 1028 feet; the oil-sand consisting of two beds 18 feet thick, separated by 7 feet of slate. Considerable gas was obtained in this well, from the top of the oil-sand down to a depth of 1091 feet. Oil was also struck, which Mr. O. P. Taylor informed me personally would have amounted to five or six barrels a day if properly pumped. This was not considered sufficient to make the well a paying and profitable one and it was abandoned.

In January, 1879, Taylor's third venture, known as the Wycoff well, was completed, on north middle lot No. 36, Alma township. It was 1300 feet deep and the oil-sand, 59 feet thick, was struck at a depth of 1212 feet. This well was dry. The next, located on the Crandall farm in lot No. 4, Alma township, and known as the Triangle No. 1, was completed July 4th, 1880. The elevation of the top of this well is 1825 feet; the oil-sand struck at 1109 feet was of a superior quality to that struck in the other wells, and after being torpedoed the well filled up within an hour with 700 feet of oil. A careful record of this well was kept, as follows:

Well-mouth above ocean in feet,			1825
1. Clay, sand and gravel,	100 to	100	= 1725
2. Dark gray shale,	30 to	130	= 1695
3. White sandstone and shale,	40 to	170	= 1655
4. Red shale and sandstone,	15 to	185	= 1640
5. Chocolate shale,	5 to	190	= 1635
6. Red sandstone and shale,	16 to	206	= 1619
7. Chocolate shale and sandstone,	4 to	210	= 1615
8. Gray sandstone containing water,	8 to	218	= 1607

9. Gray sandstone,	12 to	230 =	1595
10. Red sandstone,	6 to	236 =	1589
11. Gray slate,	30 to	266 =	1559
12. Gray shale,	14 to	280 =	1545
13. White shale and sandstone,	3 to	283 =	1542
14. Gray shale,	4 to	287 =	1538
15. Gray sandstone,	4 to	291 =	1534
16. Dark gray sandstone,	7 to	298 =	1527
17. Gray slate,	30 to	328 =	1497
18. Light gray shale,	20 to	348 =	1477
19. Gray slate containing sand shales,	21 to	369 =	1456
20. Light gray slate,	79 to	448 =	1377
21. Gray shale, containing fragments of fossils,	4 to	452 =	1373
22. Soft gray slate,	31 to	483 =	1342
23. Argillaceous sandstone,	22 to	505 =	1320
24. Gray shale,	30 to	535 =	1290
25. Gray shale containing fragments of fossils,	4 to	539 =	1286
26. Red shale,	1 to	540 =	1285
27. Gray slate,	52 to	592 =	1233
28. Gray shale containing fossil remains,	4 to	596 =	1229
29. Gray slate,	21 to	617 =	1208
30. Gray shale, containing fossil remains,	1 to	618 =	1207
31. Soft gray shale,	47 to	665 =	1160
32. Gray sandstone,	40 to	705 =	1120
33. Dark gray shale and slate,	80 to	785 =	1040
34. Gray slate, containing fragments of fossils,	61 to	846 =	979
35. Gray sandy shale, containing fragments of fossils,	9 to	855 =	970
36. Gray shale,	120 to	975 =	850
37. Gray sandstone containing oil and salt water,	20 to	995 =	830
38. Gray shale,	114 to	1109 =	716
39. Soft gray sandstone, top of oil sand,	17 to	1126 =	699
40. Harder gray sandstone,	17 to	1143 =	682
41. Soft gray sandstone, bottom of oil-sand,	10 to	1153 =	672
42. Gray shale and slate,	24 to	1177 =	648
Total depth of well,			1177 feet.

The top of this well is 625 feet below the bottom of the Olean conglomerate, making the distance between the top of the Allegany oil-sand in this well and the Olean conglomerate 1734 feet. The lower 525 feet of this interval of 625 feet is occupied by gray shale and slate and sandstone; above this occurs the sub-Olean conglomerate, which is the middle member of the Pocono sandstone, No. X., ranges from 30 to 40 feet thick, and occurs immediately below the gray shale representing the upper part of the Pocono sandstone, and the red shale representing the Mauch Chunk red shale, No. XI. This section, together with that of the record of the well, can be taken as a representative section of the rocks in the Allegany oil- and

Name of Well.	Location of Well.	Elevation of Well above Tide.	Depth to Top of Oil-Sand.	Thickness of Oil-Sand.	Elevation of Top of Oil-Sand above Tide.
	Scio Township.				
Riley Allen, No. 1	Lot No. 2 in Allentown	1805	980	50	825
Allen & Noble	Lot No. 2, Ketchum farm	1990	1188	47	802
Neff	N. W. corner, lot No. 1	1910	1117	25	673
	Alma Township.				
Homestead, No. 1	Lot No. 1 near Allent'wn R.R. sta.	1850	1044	48	806
Triangle, No. 1	Crandall farm lot No. 4	1825	1109	...	718
Royal Oil Co.'s, No. 2	Whitter " " "	1860	1187	...	673
" " " " 1	" " " "	1885	.....	...	.....
Triangle, No. 3		1970	1204	35	766
Wycoff	North middle lot, No. 26	1980	1212	59	768
Central	" " " 6	1950	1218	60	732
Pikeville, No. 1		1775	1028	18 sand, 7 slate, 18 sand	747
" " 2		1875	1118	30	757*
Duke & Norton, No 1	Lot No. 23	2160	1404	19	756
" " 3	N. W. corner, lot No. 39	2170	1433	37	737
" " 7	Russell farm, lot No. 22	2060	1308	19	752
" " 9	South side " 40	2285	1649	24	636
McCalmont Oil Co., No. 1	Johnson farm, lot No. 3	.....	1387	44	.....
" " " 2	" " "	.....	1338	49	.....
" " " 4	" " "	.....	1285	54	.....
	Wirt Township.				
Taylor & Humphrey, No. 1	Phillips farm, lot No. 17	.....	1490	20	.....
" " " 2	East of lot No 1	.....	1458	18	.....
" " " 3	North of Nos. 1 and 2	.....	1471	21	.....
Flanigan & Cheeseman " 1	Griffen farm, lot No. 41	1650	855	20	795
" " " 2	" " 41	1655	851	4, 10, 12, 18	804
" " " 3	" " 41	1635	834	4, 10	801
" " " 4	" " 41	1630	827	32	798
Flanigan & Sanders, No. 2	Sanders farm, lot No. 41	1650	852	38	798
" " " 3	" " 41	1652	839	11	813
	Bolivar Township.				
McCalmont Oil Co., No. 7	Reed farm, lot No. 56	.....	819	29	.....
" " " 8	" " 56	.....	820	32	.....
" " " 9	" " 56	.....	840	44	.....
" " " 10	" " 56	.....	1120	27	.....
" " " 11	" " 56	.....	809	30	.....
D. P. Taylor	William farm, lot No. 37	1920	1208	40	712
Campbell	Southport farm, lot No. 16	1980(?)	1196	35	784
Fisher, No. 1	Miller " " 31	1790	965	30	825
" " 2	Caudell " " 31	1795	1362(?)	18	813 (?)
Klinger, Plumber & Moran	Morse " " 37	1985	1229	8	756
Mutual Oil Co., No. 6	Richardson " " 37	1985	1224	14	761
" " " 7	" " " 37	2000	1240	13	760
" " " 1	" " " 37	1940	1167	22	773
	Wirt Township.				
Richburgh, No. 1		2000	1208	20	792
Morse and Williams	Smith farm, lot No. 20	1720	1140	...	580
	Bolivar Township.				
Pliny Parker	Stillman farm, lot No. 25	1795	1300†	...	495
Willett	Mills farm, lot No. 33	1790	1302†	...	488
Waugh & Porter, No. 1	" " 34	1880	1330†	...	500
" " " 2	" " 34	2000	1315†	...	585
Bradley & Co. Gas	" " 28	1785	1062	...	723
Charring Oil Co., No. 8	Richardson farm, lot No. 37	1765	996	...	769
Northrup & Co., No. 1	Lot No. 68	.....	1445	23' 6"	.....
" " 2	.....	.....	1455' 6"	"	.....
" " 4	.....	.....	1299' 6"	"	.....
" " 6	.....	.....	1290	"	.....
	Genesee Township.				
Cranston & Co., No. 1	Hatch farm, lot No. 29	2225	1632	...	593
" " 2	" " 29	2235	1655	...	580

* Elevation of lower Waugh & Porter sand, 425.

† Upper Waugh & Porter sand.

Name of Well.	Location of Well.	Elevation of Well above Tide.	Depth to Top of Oil Sand.	Thickness of Oil Sand.	Elevation of Top of Oil Sand above Tide.
	Genesee Township.				
Willetts, No. 4	N. E. corner, lot No. 3	2085	1308	28	727
" " 19		2125	1402	20	723
" " 25		1980	1259	25	721
Willett	Lot No. 22	.....	1122	...	.....
J. W. Davis, No. 1	S. E. corner, lot No. 31	2085	1410	30	625
" " 2	450′ north of No. 1	1975	1328	34	647
" " 3	" " 2	1920	1256	39	664
" " 4	" " 3	1925	1251	34	674
" " 5	" " 4	1880	1197	30	683
" " 6	" " 5	1810	1108	40	702
" " 7	" " 6	1760	1053	42	707
" " 8	400′ west of No. 7	1768	1058	27	710
Davis & Haldeman, No. 2	Lot No. 24	1675	914	22	761
" " " 3	"	1755	995	20	760
" " " 4	"	1820	1045	28	775
" " " 5	"	1860	1108	14	752
" " " 6	"	1845	1096	15	749
" " " 7	"	1790	1040	35	750
Armour & White, No. 2	Hill farm	1840	1093	35	747
" " 4		1735	959	25	776
" " 5		1785	1017	14	768
" " 6		1765	995	12	770
" " 7		1735	963	27	772
" " 8		1715	955	19	760
	Clarksville Township.				
Davis & Haldeman, No. 8	Lot No. 17	1785	1064	15	721
" " " 9	"	1835	1100	(15 sand) (5 slate) (15 sand)	735
" " " 10	"	1770	1058	10	712
" " " 11	"	1850	1103	(15 sand) (29 slate) (10 sand)	747
" " " 12	"	1915	1182	10	733

gas-district. An outcrop of the sub-Olean conglomerate may be seen facing the Genesee river a mile and a half north of Wellsville; the elevation of the rock at this point is 2160 feet. Another outcrop, the elevation of which is about 2000 feet, may be seen facing the Genesee river 6 miles south of Wellsville.

During 1880 there were 50 wells drilled in the Allegany oil-field, 42 of which contained either gas or an unprofitable amount of oil; 8 wells produced oil at an average of 9½ barrels per well. It was not until 1881 that the oil-trade attached any importance to the oil-developments in Allegany County. To Mr. O. P. Taylor more than to any one else is due the credit of advancing the oil- and gas-interests of the Allegany district.

The general geological structure of the Allegany oil- and gas-district is clearly defined by the selected well-records on pp. 932 and 933, the geographical position of which can be ascertained by referring to the corresponding township and lot number on the map.

These records show that the top of the Allegany oil- and gas-sand

Name of Well.	Location of Well.	Length of Casing.	Depth to Top of Oil Sand.	Depth to Bottom of Oil Sand.	Total Depth of Well.	Remarks.
Gilchrist, No. 1.....	Merritt farm...		1180	1200	1219	
Green, " 1.....	Green " ...	387	1441	1455	1471	
" " 2.....	" " ...	290	1307	1315	1358	Casing pulled out and plugged.
" " 3.....	" " ...	405	1492	1514	1534	
" " 4.....	...		1417	1456	1471	Casing pulled out and plugged.
Stillman " 1.....	Stillman farm	497	1030	1047	1068	
" " 2.....	" "	405	1132	1146	1163	Casing pulled out.
" " 3.....	" "	447	1005	1023½	1012	"
" " 4.....	" "	403	1119	1135	1152½	
Young " 2.....	Young "	400	1500	1515	1530	
Hardison " 5..... ...	Merritt "		1216	1259	1275	Gas struck at 1200'.
" " 6.....	" "	283	1284	1301	1340	" " 1238'.
Fisher " 1.....	Fisher "		1196			
" " 2.....	" "		1268			
" " 3.....	" "		1452			
" " 4.....	" "		1437			
" " 5.....	" "	842	1073	1098	1118½	Gas struck at 1055'. [and 1161'.
" " 6.....	" "		1134	1167	1179	" " 1112', slate bet. 1146'
" " 7.....	" "		1304	1324½	1335	" " 1269'.
Anderson " 1.....	Woodward "		1019	1046	1065	
" " 2.....	" "		1017	1027	1067	Gas struck at 953'.
" " 3.....	" "	490	1224	1238	1256½	" " 1162' through 1216'.
Labar " 1.....	Labar "		1255	1316	1365	
		360c'g				
Adams, " 1.....	Adams "	57 dr. pipe	1117	1138	1158	
Wiser & Kinkaid, No. 2	Ford "		1340	1392	1415	Gas struck at 1260' to 1280'.
Chase & Mallory, " 1	" "			1280	1281	
" " 2	" "			1321	1356	
Jordan, No. 1.....	Jordan h'rs "		920	940	973	
Wells, " 1.....	Wells "	492	1665	1688	1714	
Stives, " 1.....	Stives "	340	1845		1605	Gas struck at 1500'.
			1st 1094	1110		
Smith, " 1.....	Springer "	300	2d 1129	1150	1165	
			1121	1137		
" " 2.....	" "	310	1146	1173	1192	
			1160	1172		
" " 3.....	" "	315	1179	1210	1234	
" " 4.....	" "	300	1170	1201	1241	
			1196	1216		
" " 5.....	" "	290	1229	1237	1259	
			1st 1064			
			2d	1090		
" " 6.....	" "	405	1119	1126	1175	
			3d 1148	1157		
Fitz, " 1.....	Fitz "		1272		1324	Gas struck in sand at 1238.
Wells, " 2.....	Wells "		1640	1662	1685	96 qt. torpedo exploded in sand. 478 ft. deep.
" " 2.....	" w'ter well					
Howe, " 3.....	Howe farm.....		1338	1366	1400	Slate, 1327 to 1332, 40 qt. torpedo exploded sand.
Lesker, Jordan & Co. No. 1.....			1271	1289	1319	Gas at 1187, slate at 1206, gas from 1232 to 1247, shot with 40 qt. torpedo.
Campbell Oil Co. Sawyer No. 3, or Campbell No. 8.	Ira Sawyer farm		1272	1295	1326	Gas at 1200.
Chase & Mallory, No. 1		286	1230	1248	1286	
" " 1		250	3d 1280	1320	1356	
			gas	to		
R. Allen. " 2		338	sand 1222	1246	1350	
Jordan Heirs, " 2		304				
Hogan, Murphy & Basele, No. 1.....		400			1018	Struck gas at 963, oil at 981—18 ft. gas sand, 18 ft. oil sand.
Hogan, Murphy & Basele, No. 2.....		370			1012	Struck gas at 947, oil at 970—23 ft. gas sand. 14 ft. oil sand.
Hogan, Murphy & Basele, No. 3.....		378			1006	Struck gas at 946, oil at 966—20 ft. gas sand, 18 ft. oil sand.
Hogan, Murphy & Basele, No. 4.....		410			1054	Struck gas at 988, oil at 1020—22 ft. gas sand, 12 feet oil sand.

is extremely undulating, and in many localities has excessive dips. For instance, the dip of the sand between the Davis and Haldeman's wells and the Davis well No. 1 is at the rate of 160 feet to the mile. The greatest dip which I have ever observed in the Bradford sand within the main Bradford district is between Tarport and State Line, 18 feet to the mile. This comparatively heavy and variable dip in the Allegany district, I think has been the cause of confusing the geology of the district, as compared with that of Bradford, in the minds of both geologists and operators.

The topography of the top of the Allegany oil- and gas-sand may be more fully appreciated by designating on the map (Plate III) the elevation above tide of the sand at each well included in the above list. In addition to this list of wells, the locations of which are all directly referable to the map, I have compiled a list of a few of the other well-records in my possession in order to show especially (1) the depths at which all surface fresh water is cased off; (2) the thicknesses in different localities of the oil and gas-sand; and (3) the depth to which the wells have generally been drilled below the producing sand. These records are noted in the preceding table (p. 934).

Most of the gas-wells in the Allegany district are at present controlled and operated by the Empire Gas and Fuel Company, Limited, of Wellsville. In the spring of 1887, this company had about 2000 individual consumers along its various pipe-lines, which measured in the aggregate about 125 miles, the pipes ranging from 8 inches to 2 inches in diameter. At the same time the company owned 102 wells, 73 of which produced oil, 4 oil and gas combined, and 25 gas alone. These wells are enumerated in the following list. No measurement has ever been made of the actual production in cubic feet of any of the gas-wells in the district, but the relative productive capacity of the gas-sand in different parts of the field is shown by the maximum pressure of the gas which will accumulate in the wells noted in the table.

No.	Lot.	Township.	Farm.	Character of Well.	Highest Confined Pressures at gas-wells. Pounds.	In Minutes.
1	11	Genesee,	Dancy,	Gas,	50	—
2	21	"	Bullock,	Oil,	—	—
3	21	"	"	"	—	—
4	21	"	"	"	—	—
5	14	"	Young,	"	—	—
6	14	"	Green,	"	—	—
7	14	"	"	"	—	—
8	7	"	Stullman,	"	—	—

No.	Lot.	Township.	Farm.	Character of Well.	Highest Confined Pressures at gas wells. Pounds.	In Minutes.
9	7	Genesee,	Stillman,	Oil,	—	—
10	7	"	School House,	"	—	—
11	16	"	Merritt,	"	—	—
12	16	"	"	"	—	—
13	16	"	"	"	—	—
14	16	"	"	"	—	—
15	16	"	"	"	—	—
16	9	Clarksville,	"	"	—	—
17	9	"	"	"	—	—
18	9	"	"	"	—	—
19	17	"	Springer,	"	—	—
20	17	"	"	"	—	—
21	17	"	"	"	—	—
22	17	"	"	"	—	—
23	17	"	"	"	—	—
24	17	"	"	"	—	—
25	3	"	Adams,	Gas,	90.7	—
26	3	"	McJordan,	"	90.7	—
27	2	"	Painter,	"	90.7	—
28	2	"	Lovell and Willett,	"	90.7	—
29	59	Wirt,	Lebar,	"	100	21
30	60	"	Isaiah Jordan,	"	100	16
31	61	"	Deyoe,	"	130	—
32	51	"	W. Jordan,	"	95	25
33	51	"	"	"	125	20
34	58	"	Ballard,	"	85	17
35	50	"	Evans,	"	70	10
36	49	"	Barton,	"	70	—
37	51	"	Jordan heirs,	"	110	7
38	51	"	"	"	117	Shut in.
39	43	"	Knox,	Oil and gas,	—	—
40	43	"	"	"	100	3
41	43	"	"	"	100	3
42	34	"	Richardson,	Gas,	140	15
43	34	"	"	"	140	—
44	34	"	Lawrence,	"	150	17
45	26	"	Brown,	"	Drowned out by oil.	
46	26	"	Patterson,	"	100–100	—
47	30	Bolivar,	J. Goodrich,	Oil,	—	—
48	30	"	"	"	—	—
49	30	"	"	"	—	—
50	30	"	"	"	—	—
51	30	"	George Beers,	Gas,	Drowned out by oil.	
52	22	"	Wakeman,	"	125	—
53	30	"	Goodrich,	Oil,	—	—
54	14	"	Fitzgerald,	"	—	—
55	14	"	Sawyer,	"	—	—
56	15	"	Anderson,	"	—	—
57	15	"	"	"	—	—

No.	Lot.	Township.	Farm.	Character of Well.	Highest Confined Pressures at gas wells. Pounds.	In Minutes.
58	15	Boliver,	Anderson,	Oil,	—	—
59	15	"	Sawyer,	"	—	—
60	15	"	"	"	—	—
61	15	"	"	"	—	—
62	15	"	"	"	—	—
63	8	"	Ford,	"	—	—
64	8	"	Dunham,	"	—	—
65	8	"	"	"	—	—
66	8	"	"	"	—	—
67	16	"	Wetherby,	"	—	—
68	16	"	"	"	—	—
69	16	"	"	"	—	—
70	16	"	"	"	—	—
71	16	"	"	"	—	—
72	16	"	"	"	—	—
73	16	"	"	"	—	—
74	16	"	Howe,	"	—	—
75	16	"	"	"	—	—
76	16	"	"	"	—	—
77	8	"	Wiser and Kincaid,	"	—	—
78	8	"	"	"	—	—
79	8	"	Dunham,	"	—	—
80	8	"	"	"	—	—
81	8	"	Ford,	"	—	—
82	8	"	"	"	—	—
83	8	"	S. Wetherby,	"	—	—
84	8	"	"	"	—	—
85	8	"	"	"	—	—
86	8	"	Glycerine Lot,	"	—	—
87	1	Alma,	Noff,	Oil and gas,	—	—
88	1	Wert,	Drumb,	Oil,	—	—
89	1	"	Riley Allen,	Gas,	—	—
90	1	"	"	"	110	—
91	9	"	Wright,	Oil,	100	—
92	24	Bolivar,	Phillips,	"	—	—
93	24	"	"	"	—	—
94	24	"	Wright,	"	—	—
95	24	"	"	"	—	—
96	24	"	"	"	—	—
97	24	"	Crandall,	"	—	—
98	24	"	"	"	—	—
99	24	"	"	"	—	—
100	24	"	"	"	—	—
101	3	Alma,	Pike,	"	—	—
102	3	"	"	"	—	—

The peculiar character of the gas-sand in the Allegany district is such that I believe we can safely anticipate that no sudden termi-

nation will come to the production of gas, but that the decline will be more gradual than the decline of other gas-districts which draw their supplies from coarser and more open gas-rocks.

The prices charged the consumers by the Empire Gas and Fuel Company, Limited, are as follows:

Cook stoves, ...	$3.50 per month.
Heating stoves, according to the size of the room, ranging from	$3 to $6 per month.
Lights in dwellings, ...	$0.25 per month.
Lights in stores, ...	$0.30 per month.
Boilers, from	$1.25 to 1.50 per day.
Furnaces, from	$6 to $9 per month.

Although the Allegany gas-wells differ in many respects from those of Northern Pennsylvania, yet in the main the history of these wells will no doubt be found to be much the same as the history of the gas-wells in Northern Pennsylvania, particularly those in northern McKean County.

Oil- and Gas-Sands of Pennsylvania.

An interesting practical question relating to the geology of the Allegany oil- and gas-district is the relationship of the Allegany sand to the oil- and gas-sands in other regions, particularly those in Pennsylvania, southwest as far as Pittsburgh. The strata so far found to contain natural gas in commercial quantities between Allegany County, New York, and Washington County, Pennsylvania, occur within a vertical range of about 3000 feet, are all of the Carboniferous and Devonian group of rocks, and extend from the Mahoning sandstone, at the base of the Lower Barren Coal Measures, and, on an average, about 500 feet below the great Pittsburgh coal-bed, down to the Roy and Archer oil- and gas-sand in Elk county. A small show of gas is found in Greene county, Pennsylvania, in two sandstones, one 165 feet and the other 425 feet respectively below the Pittsburgh coal-bed. These sandstones are above the Mahoning sandstone referred to; but since they do not produce gas in commercial quantities they are not considered here. Next below the Mahoning sandstone, gas has been found in Washington County, in what is known as the Homewood sandstone, a member of the Pottsville Conglomerate No. XII., which lies at the base of the Lower Productive Coal Measures. The next rock below the Homewood which has produced gas in commercial quantities is the Pit

Hole Grit, a representative of the Berea Grit of the Ohio Survey. This sandstone is about 1725 feet below the Pittsburgh coal-bed, and between 800 and 900 feet below the Homewood sandstone.

Next below the Berea Grit, and immediately below the Pit Hole Grit (which rocks are probably the same) occur the sands of the Venango-Butler oil- and gas-sand group, which are representatives of the Catskill Old Red sandstone strata. In the upper part of this group is located the great Murrysville gas-sand, from which almost the entire gas-supply of Pittsburgh is derived. This group varies from 300 to 600 feet thick, being thinner in the northern part of Pennsylvania, and thicker in the southwestern corner of the State; next below this group occur from 300 to 350 feet of shales of the Chemung, which produce no gas. Below this barren interval are the Cherry Grove oil-sand and the Sheffield gas-sand of southern Warren County, which occur about 200 feet below the base of the Warren group; from 300 to 400 feet below the Warren group lies the Bradford or Allegany sand. The only sands which have so far been found to produce gas below the Bradford or Allegany gas-sand are the Cooper sand in Warren County, which, however, is probably one of the members of the Bradford sand, or the upper oil-sand of the Waugh and Porter well in Allegany County, N. Y., and the Roy and Archer well sand in northern Elk County. This latter sand occurs about 500 feet below the Bradford sand and about 1800 feet below the geological horizon of the Murrysville sand. All the gas-sands above referred to are in the Portage and Chemung formations. No clear division-line between these two formations has ever been established.

The geographical relationship of the Allegany district to the oil and gas-districts of Western Pennsylvania and the geological relationship of the oil- and gas-sands from Allegany County southwest to Washington County, Pennsylvania, are shown by the map and columnar sections on the accompanying chart (Plate II).

Largest Producing Gas-Well in New York.

The greatest amount of gas which was ever discharged per diem from any well in New York State was from the McMullen & Hallock gas-well, commonly known as the "Mullen snorter." This well is located on the Loup farm, section 1, in the extreme southwest corner of Olean township, Cattaraugus County, and in an air-line 5½ miles south-southwest of the village of Olean. Gas was struck in this well on the 30th of May, 1877. The elevation of the

top of the well above tide is 1785 feet. Solid rock was struck at a depth of 16 feet, and at a depth of 196 feet all the surface-water was cased off. The strata above the depth of 625 feet were composed of gray shales and slate, with occasional thin seams of sandstone. From 625 feet to 675 feet a fine-grained "shelly" sandstone 50 feet thick was pierced. Below this sandstone, and to a depth of 870 feet, strata were passed through similar in character to those in the other part of the well. From 890 to 960 feet a fine-grained sandstone, alternating with shale, was drilled through. Gas was first encountered in the well at a depth of 1180 feet in a sand-bed which was 25 feet thick. The Bradford oil- and gas-sand proper was struck at a depth of 1230 feet. I visited this well June 1st, 1877, and from careful measurements estimated the discharge of gas to be at the rate of 17,000 cubic feet per minute, or 24,480,000 cubic feet per twenty-four hours. There was discharged in conjunction with the gas about a barrel of oil per day. The top of the oil-sand in this well is 1785 feet below the bottom of the Olean Conglomerate, which outcrops at the famous Olean Rock City. The gas-sand in the vicinity of this well is nearly horizontal, having only a dip of about 11 feet per mile in a direction south 15° west. The discharge of gas continuously decreased until 1882. Since that date the well has produced from two to three barrels of oil per day, with a comparatively small amount of gas. This well is interesting for several reasons: First, it is the largest gas-well ever found in the State of New York, and one of the largest in the Bradford oil-pool; second, the well, although a large gas-producer, is not located on an anticlinal, but in a synclinal; third, on account of the intimate association of oil and gas in the well, and the short life of the gas-flow; fourth, this well produced more gas per day than any well whose discharge had ever been measured up to June, 1877. There is no doubt that larger gas-wells had previously been obtained in Pennsylvania, but the discharge of none of the larger Pennsylvania wells had ever been actually measured.

Explorations for Gas in Central New York.*

Professor Charles S. Prosser, Assistant in the Geological Laboratory at Cornell University, has for some time been engaged in obtaining as far as possible a record of the test-wells drilled in New

* These notes and also those relating to natural gas in Ontario County have been kindly furnished by Professor Prosser for original publication in this place.

York outside of the Allegany oil- and gas-field, with the view not only of ascertaining the facts connected with the discovery of natural gas, but also of determining the thickness of the different geological formations. These facts Professor Prosser is tabulating with the expectation of using them as a basis for a paper on the stratigraphical geology of New York State. They have been gathered principally in the central part of western New York. He has furnished them to me with such brief references as may make them intelligible for publication in this connection.

Record of Well Drilled at Ithaca, Tompkins County. Location, in the Valley, One-Quarter Mile South of Ithaca.

	Well-mouth above ocean in feet,		396
	1. Lower Portage shales,	340 to 340	+ 56
	2. Genesee shale,	100 to 440	— 44
	3. Tully limestone,	30 to 470	— 74
	4. Hamilton group; composed of argillaceous and arenaceous shales, thin sandstones with calcareous shales and sandstones,	1142 to 1612	— 1216
	5. Marcellus shale,	82 to 1694	— 1298
	6. Corniferous limestone,	78 to 1772	— 1376
	7. Oriskany sandstone,	13 to 1785	— 1389
	8. Lower Helderberg limestones,	115 to 1900	— 1504
Salina or Onondaga Salt Group.	9. Shale,	344 to 2244	— 1848
	10. Rock Salt,	24 to 2268	— 1872
	11. Shale,	6 to 2274	— 1878
	12. Rock Salt,	54 to 2328	— 1932
	13. Shale,	12 to 2340	— 1944
	14. Rock Salt,	17 to 2357	— 1961
	15. Shale,	31 to 2388	— 1992
	16. Rock Salt,	21 to 2409	— 2013
	17. Shale,	67 to 2476	— 2080
	18. Rock salt,	42 to 2518	— 2122
	19. Shale,	24 to 2542	— 2146
	20. Rock salt,	48 to 2590	— 2194
	21. Shale,	82 to 2672	— 2276
	22. Rock salt,	42 to 2714	— 2318
	23. Green shale,	308 to 3022	— 2626
	24. Mottled red and green shale,	6 to 3028	— 2632
	25. Green shale,	102 to 3130	— 2734
	Depth of well (December 15th, 1887, in Salina shale),		3130 feet

The total depth of the well, as given by Mr. Rust, the contractor, is 3185 feet. The record stops at 3130 feet.

The largest amount of gas was obtained from the last 30 feet of

the Genesee shale and the first 50 feet of the Hamilton shales, below the solid 30 feet of Tully limestone. When the gas was lighted, the flame reached from 4 to 8 feet in height. There was no particular increase in the amount of gas when the Marcellus shales were reached. A small amount of gas with a very disagreeable odor was obtained in the Lower Helderberg limestones.

Record of Well Drilled at Clyde, Wayne County. Location, in Clyde Village. Altitude, approx., 389′ A. T.

Salina Group.	1. Gray, green and blue marls with gypsum,	152 to	152 = +	246
	2. Red marls,	156 to	308 = +	90
	3. Blue and green marls,	32 to	340 — +	58
	4. Dark blue limestone, upper division of Niagara group,	110 to	450 = —	52
	5. Shaly limestone, lower division of Niagara,	225 to	675 = —	277
	6. Clinton group: oölitic iron ore in sample from 675′–690′,	83 to	758 = —	360
	7. Red shale of Medina,	24 to	782 = —	584
	8. Greenish-gray sandstone,	3 to	785 = —	587
	9. Red shales alternating with red sandstone and forming the Red Medina,	915 to	1700 = —	1502
	10. Gray Medina or gray sandstone of Oswego (Vanuxem),	92 to	1792 = —	1594
	11. Depth of well (in Gray Medina),			1792 feet

At 380′, in the blue Niagara limestone, some gas was obtained, and so on to about 500′. The flow of gas has continued the same during the two months that have passed since it was found. At 685′ a "pocket" of gas was developed in the Clinton which was soon exhausted. The gas, when lighted, supports a flame 3 or 4 feet high.*

* Mr. George O. Baker, Secretary of the Clyde Mining Company, under date of February 21, 1888, furnishes the following facts relating to the Clyde exploration well. Mr. Baker has grouped the strata passed through from observations made by Professor Prosser.

"A few facts regarding the well may not be uninteresting. A corporation was organized in May, 1887, with a capital of $3000. In the summer a contract was made, and drilling commenced on the vacant lot near the glass works, September 13th. Plaster rock was reached at 20 feet; this, with gray, green and blue marls, continued until at 152 feet red shales were struck, and these continued until 108 feet from the surface had been reached, then 32 feet of light blue and greenish marl. These formations are understood to be the lower part of the salina or Onondaga salt group. At 340 feet the Niagara limestone was reached and continued to 675 feet; the upper 110 feet being dark blue limestone of the upper division of the Niagara,

Record of Well Drilled at Wolcott, Wayne County. Location in the Ravine at Wolcott Village. Altitude, approx., 317′ A T.

1. Shaly layers of Niagara limestone above and Clinton below, thickness of shales below top of latter not determined, . .	214 to	214 = +	103
2. Oölitic iron ore and shales of Clinton, group, ...	16 to	230 = +	87
3. Red shale alternating with red, siliceous sandstone, forming the Red Medina, .	690 to	920 = —	603
4. Gray Medina or Oswego sandstone of Vanuxem, ...	210 to	1130 = —	813
5. Some blue shale alternating with gray siliceous sandstone, similar to the Oswego sandstone, ...	170 to	1300 = —	983
6. Undoubted Hudson River blue shale, .	55 to	1355 = —	1038
7. Gray sandstone containing gas, followed by dark blue shale, ...	5 to	1360 = —	1043
8. Lower part of Utica shale, . . .	590 to	1950 = —	1633
9. Compact blue limestone alternating with shaly layers forming top of Trenton, .	750 to	2700 = —	2383

Gas was found at 2092 and at 2330, the larger amount being found at the latter depth.

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and the lower 225 feet the shaly limestone of the same Niagara group. At 675 feet, the Clinton group was reached, and continued to 758 feet, the upper 15 feet being a greenish shale with traces of iron, and the balance a dark gray limestone. At 758 feet the 'Red Medina' was reached, and this, with varying color, continued to 1700 feet. At this depth the Gray Medina or Oswego sandstone was reached, and the drilling was continued in this to the depth of 1792 feet, when operations were suspended, November 23d, 1887.

"Complete samples of the drillings have been furnished to Prof. Charles S. Prosser, of Cornell University, and much valuable information has been received from him. The specimens are in the Geological Laboratory of the University, and a complete set of samples will be furnished for the laboratory of the Clyde High School as soon as the same can be properly arranged.

"At 360 feet gas was indicated, and at 380 feet quite a flow was obtained. At 685 feet a 'pocket' of gas was struck, which, for a short time, increased the flow perceptibly. Not much change has occurred in the flow since drilling has ceased; at least, there has been no decrease.

"At 110 to 120 feet the same mineral water was found, which is somewhat common here, apparently the same as at the old spring by the river, at Streeter's malt house and at the glass factory.

"At 175 feet a very strong salt water was found, also in a crevice at the depth of 340 feet (top of Niagara), salt water was found.

"The well is thoroughly cased to the depth of 352 feet, 11 inches with extra heavy casing, completely shutting off the water, and from thence to the bottom is what may be called a 'dry hole.' It is to be hoped that enough interest may be taken to furnish the needed money to reach a depth of 3000 feet, which will sufficiently penetrate the Trenton, the famous gas-bearing rock, to demonstrate whether gas in paying quantity can be found in Clyde."

The upper portion of the Trenton limestone in the Wolcott well, and probably for a considerable distance both east and west of this village, is a very compact rock. Deeper the limestone becomes more shaly and in this part of the formation the gas was obtained. The great difference in the structure of the western and eastern Upper Trenton limestone is readily seen when samples of the drillings from Ohio or Indiana wells are compared with those from this well. The porosity of the western Trenton seems to be almost entirely wanting in this part of New York.

Mr. H. W. Hatch, of Richburgh, who drilled the Wolcott well, states that when the well was finished in the latter part of October, 1887, the well produced, by meter measurement, 5000 cubic feet of gas per day; on December 1st, the production remained the same.

Some gas was obtained from 1100 feet on; the largest amount came from the sandstone at 1355 feet in the Hudson River group. Gas was got in the Trenton limestone first at 2092 feet, and the largest quantity at 2330 feet. Pieces of the limestone were thrown out of the well from the 2330 feet "pocket" and it is claimed that the flame of the gas when lighted, was from 15 to 20 feet high.

From a partial record of a well drilled in Ontario, Wayne County, New York, I am inclined to think that the bottom of the Wolcott well is within about 100 feet of the bottom of the Trenton series, below which is 100 feet of Calciferous limestone before the Archæan can be reached.

Record of State Well at Syracuse, from the Report of the Superintendent of the Onondaga Salt Springs, N. Y. Assembly Doc., No. 32, for 1885, page 15.

Salina.	1. Drift, 2. Red marls,	. . . . .	555 to 555 = —	155
	3. Variegated marls and limestones,	. .	23 to 578 = —	178
	4. Niagara limestone,	. . . . .	342 to 920 = —	520
	5. Clinton shales and iron-ore,	. . .	105 to 1025 = —	625
	6. Transition shales,	. . . . .	50 to 1075 = —	675
	7. Red brown sandstone of the Medina,	.	100 to 1175 = —	775
	8. Red sandstone alternating with gray and brown sandstones,	. . . .	640 to 1815 = —	1415
	9. Gray Medina or sandstone of Oswego,	·	154 to 1969 = —	1569
	10. Bottom of well in Gray Medina,	. . . . .		1969

An examination of the records of the Wolcott, Clyde and State wells shows a considerable variation in the thickness of the Red Medina. It is 690 feet thick in the Wolcott well, 942 feet in the Clyde and from 740 feet to possibly 810 feet in the State well at

Syracuse. A difference of 252 feet in the thickness of the Red Medina in the Wolcott and Clyde wells, which are on nearly a north and south line, shows a considerable thickening of this formation to the southward. The diminished thickness of the Red Medina in the State well, thirty miles east of Clyde, agrees with the known thinning of the Medina to the eastward.

Southward from Ithaca to Pennsylvania, the thickness of the formations can only be estimated. This portion of the geological series includes the three following formations in ascending order from the Genesee Shale and Lower Portage of H. S. Williams: The Ithaca group of Vanuxem, Upper Portage of H. S. Williams and the Chemung. Prof. Hall estimated the thickness of what he calls Chemung (which includes Williams' Upper Portage and Vanuxem's Ithaca group) to be 1500 feet for this section of New York (see Geol. Rept., 4th dist., p. 260); while Prof. H. S. Williams has given the thickness of the same series as about 1950 feet. (See Bulletin No. 3, U.S.G.S.)

Section from the Top of the Genesee Shale on Cayuga Lake, New York, to Ulster, Bradford County, Pennsylvania, and thence to the Barclay Coal Field. Compiled from Surveys of Prof. H. S. Williams. (Bul. No. 3, U. S. Geol. Survey.)

		Feet.
	Barclay lower coal-bed,	—
XII.	Pottsville Conglomerate,	
XI.	Mauch Chunk Red Shale,	1000
X.	Pocono Gray Sandstone,	
IX.	Catskill Red Sandstone,	
	Upper Chemung fauna in Penna. (Top at Ulster),	300
	Typical Chemung fauna (Outcropping in vicinity of State Line, bottom of Chemung Narrows, New York),	300
	Lower Chemung fauna. (Bottom outcrops at Caroline, Danby and Newfield),	600
VIII.	Upper Portage Sandstones and Shales of H. S. Williams,	600
	Middle Portage — Upper Ithaca,	200
	Middle Portage — Typical Ithaca,	100
	Middle Portage — Lower Ithaca,	150
	Lower Portage Sandstones and Shales,	250
	Genesee Shales,	—
	Total thickness,	3500

All the facts given in the above section are not clearly expressed in the Bulletin; but Prof. Williams has kindly furnished me with a diagram which has been used somewhat in the preparation of this section. The report on Bradford County (Geol. Survey, Penna. Report G.) has been of assistance in compiling that portion of the section above the Upper Chemung.

From the sections already given there has been constructed a section giving the thickness of the different geological formations, together with the total thickness of the series, from the lowest coal-measures down to the Archæan. The section commences with the lower coal of Barclay, Penna., passes through Waverly, Ithaca, Clyde and Wolcott, N. Y., under Lake Ontario and to the Archæan of Ontario. The thickness of the formations from the Barclay coal to the Ithaca shale at the top of the Lower Portage is estimated, while for the remainder of the series it has been obtained from the wells drilled at Ithaca, Clyde, Wolcott and Ontario.

Section from Barclay, Pennsylvania, through Central New York to Ontario, Canada.

		Feet.
	Barclay lower coal-bed,	
XII.	Pottsville Conglomerate,	
XI.	Mauch Chunk Red Shale,	1000
X.	Pocono Gray Sandstone,	
IX.	Catskill Red Sandstone,	
VIII.	Chemung Sandstones and Shales,	1200
	Portage Sandstones and Shales,	1390
	Genesee Shales,	100
	Tully Limestone,	30
	Hamilton Sandstone,	1142
	Marcellus Shales,	82
	Corniferous Limestone,	78
VII.	Oriskany Sandstone,	13
VI.	Lower Helderberg Limestone,	115
	Salina Shales,	1418
	Niagara Limestone and Shales,	335
V.	Clinton Shales and Sandstone,	83
IV.	Medina Red Sandstone,	942
	Medina or Oswego Gray Sandstone,	210
III.	Hudson River and Utica Sandstones, Shales and Slates,	820
II.	Trenton Limestone,	842
	Calciferous Limestone,	
I.	Potsdam Sandstone,	150
	Total thickness,	9950

The geologic horizon of the gray limestone found in the lower part of the Ontario well is in doubt. The base of the Trenton north of Lake Ontario varies from a dark blue, sometimes massive and sometimes shaly limestone, to a "pale drab limestone of fine texture." —(*Geology of Canada*, page 190.)

Below the Black River, at Madoc, Province of Ontario, which is almost directly north of the village of Ontario, New York, and also at Marmora, west of Madoc, occur shales, sandstones, and light gray limestones from 40 to 80 feet in thickness, the geologic horizon of which is uncertain.—(*Ib.*, page 182 and page 177.)

"Below Knowlton Lake (which is nearly north of Kingston) no indications of the Potsdam formation has been observed in Canada, unless 8 feet of red soft calcareous sandstone at Marmora resting on the Gneiss, and succeeded by certain beds of limestone, without observed fossils, for 30 feet upwards have been supposed to represent it."—(*Ib.*, page 100 and page 179.)

Following the Potsdam where this is present, and resting on the Laurentian series where it is wanting, there are generally met with 30 or 40 feet of strata almost destitute of organic remains, and about the same amount with a few fossils insufficient to determine the age of the strata with certainty. The first well-characterized beds which succeed these belong to the river formation.—(*Ib.*, page 119.)

Therefore, it is quite possible that part of the light gray limestone found in the Ontario well at a depth of 2810 feet, may belong to the lower part of the Trenton formation, while the lower may or may not represent the Calciferous formation.

Natural Gas in Ontario County.*

The western part of Ontario County is the only portion of central and northwestern New York that, at present, is producing natural gas in economic quantity. The wells commence in the Genesee shale or Hamilton group, and pass through the Marcellus shale to the Corniferous limestone. The gas is obtained in the lower part of the Marcellus shale, and this gas territory is in the vicinity of the now famous West Bloomfield well. This well, I am informed, was drilled in 1863–4,† gas being obtained at about 500 feet, and the Corniferous limestone having been reached at 550 feet *approximately*. In 1870 Professor Henry Wurtz† stated that the gas came from a

* Notes furnished by Professor Charles S. Prosser.

† Prof. Wurtz's article makes the date of drilling about 1866.—Am. Jour. Sci. and Arts (2), vol. xlix, p. 336.

5-inch pipe, and gave, "in a still atmosphere, a flame some 30 feet in height." Professor Wurtz was inclined to think that the report that the well yielded "about 400,000 cubic feet of gas per day," might be true.*

An attempt to conduct the gas to Rochester in wooden pipes was a failure, and the well was filled with stones, old iron, etc. After this, for some years, the flow of gas was very small, but it is increasing again, and is reported by Mr. Peter R. Reed to be sufficient for five or six stoves.

Mr. Charles Ward, Secretary of the Honeoye Gas and Mining Company, Limited, has very kindly given me some facts in reference to the business of that company which are of interest in this connection. This company has one well at the foot of Honeoye Lake, 646 feet deep to the top of the Corniferous limestone, and intends shortly to drill additional wells. Most of the gas in this well is obtained at a depth of 610 feet, and the entire gas-pressure is reported as about 90 pounds. The company has laid 3000 feet of 3-inch pipe, 3000 feet of 2-inch pipe, and about 2000 feet of 1-inch pipe through Honeoye village, and at present is supplying 26 stoves from this well at a cost to the consumers of $3.00 per month for each stove. The plant has cost the company not over $6000.

The Ontario Gas and Improvement Company has drilled several wells about five miles north of the Honeoye Lake well in West Bloomfield township. These wells reach the Corniferous limestone at a depth of about 420 feet, and the average pressure of each well is reported to be about 35 pounds. These two companies are the only ones at present developing this territory in Ontario and Livingston counties.

Mr. Hardy, the Superintendent of the Ontario Gas and Improvement Company, reports that his company has piped the villages of West and North Bloomfield in the northwestern part of Ontario County, and Honeoye Falls in the southeastern part of Monroe County. No measurement has been made of the amount of gas which the wells are producing, but they are supplying constantly 146 stoves

* Am. Jour. Sci. and Arts (2), vol. xlix, pp. 336-7. See the Geologic Dist. of Nat. Gas in the U. S., by C. A. Ashburner, p. 18. Also, in Johnson's (Revised) Univ. Cyclopedia, vol. iii, article Gas-Lighting, p. 385, by Dr. C. F. Chandler, is the following statement: "A wonderful gas-well occurs at Bloomfield, Ontario County, N. Y. It was bored to a depth of 500 feet for oil, and yields 800,000 cubic feet, daily, of 14½ candle-gas." In answer to a note of inquiry I received the following remarkable answer from Dr. Chandler: "In reply to your favor, I would say that I am not the Prof. Chandler referred to, in connection with the gas-well in Ontario County, nor have I any idea who the professor is."

and furnaces with gas. Twelve wells have been drilled, five of which were dry, producing no gas, and of the remaining seven wells each produces about the same quantity of gas. The Corniferous limestone in these wells is struck at depths ranging from 420 to 500 feet.

Mr. Ward reports some samples of petroleum from this region.

The Genesee shales overlying the Hamilton group are reported as very bituminous in Ontario County along the shores of Canandaigua Lake. Professor J. M. Clarke, describing these shales, says:

"Along the shores of Canandaigua Lake, where the strata have an excellent development, there are at the base of the group about 20 feet of bituminous arenaceous shale, containing rows of concretions of impure calcic-carbonate, this overlaid by 40 feet of densely bituminous rock having a perfect cleavage that gives it a close resemblance to a slate. This bed is so rich in organic matters that the heat from a blow of the hammer evaporates enough of the lighter hydrocarbons to produce a very strong petroleum smell. Over limited areas it loses its schistose character, becomes compact and densely rich in bituminous matter.

Petroleum has been reported from several wells in the Canandaigua Lake region. The largest amount is reported by Professor S. F. Peckham, as obtained from a well drilled on the east shore of Canandaigua Lake in 1865, by Jonathan Watson, of Titusville, Pennsylvania. It is stated that at first this well produced five barrels of oil per day.

In the vicinity of Seneca Falls, about 12 miles southeast of Clyde, a well has been drilled, the top of which is geologically located in the Upper Salina formation. The mouth of the well is less than 75 feet below the base of the light drab limestone which Professor James Hall regarded as belonging to the last division of the Salina. (Geology of New York, Part IV., p. 123.)

The facts in regard to this well are as follows:

		Feet.
Approximate elevation of well-mouth above tide, . .		385
1. Drab ash-colored impure limestones, . . .	200 to 200 =	185
2. Blue marls, with a few streaks of red-green shale, .	100 to 300 =	85
3. Greenish-gray marl, ...	400 to 700 = —	315
4. Red shale, containing a few layers of mottled red and green shale, ...	250 to 950 = —	565
5. Dark-blue limestone, Niagara and Clinton formations, ...	400 to 1350 = —	965
6. Red shale and sandstones of Medina formation, .	150 to 1500 = —	1115
Total depth of well, ...		1500

No rock-salt was found in this well and no gas was encountered in the Niagara limestone. The gas which was found in the Medina sandstone at the depth of 1455 feet showed a pressure of 105 pounds per square inch, after the gas had been confined in the well two hours. The gas when allowed to escape from the pipe, after being confined this length of time, when lighted produced a flame 50 feet in height. According to Mr. H. W. Hatch, the well produces 12,000 cubic feet of gas per day.

A well was drilled for natural gas at Morrisville, Madison County, New York, to the depth of 1889 feet. The altitude of the well-mouth is probably more than 1200 feet above tide.

The well commences in the lower half of the Hamilton and then passes through the Marcellus shale, Corniferous limestone, Oriskany sandstone, Lower Helderberg limestone, limestones, marls, and shales of the Salina, Niagara (?) and ends in the Upper Clinton.

The united thickness of the Corniferous, Oriskany, and Lower Helderberg is 279 feet, of which 186 feet is probably Lower Helderberg. The Salina of this section consists of 325 feet of hydraulic limestone, 590 feet of shales, impure limestone and marls, closing with 225 feet of red shale. Total thickness of Salina, 1140 feet. At 1805 feet a blue argillaceous shale was encountered which may be Niagara.

Gas was obtained at 578 feet and in smaller amount at 755 feet, but not in quantity enough to be of economic value. A stratum of rock-salt, 10–12 feet thick, was reached at 1259 feet in the upper part of red and green variegated marl. Samples from 1805 and 1815 feet show traces of salt.

One of the most common questions asked is, "How deep shall we have to drill in order to reach a certain geological formation? For all northwestern New York the formation about which the inquiry is generally made is the Trenton limestone. Sufficient data are now being rapidly accumulated to enable this question to be answered for most localities. In Central and Western New York, west of Montgomery, Schoharie, Eastern Delaware and Sullivan counties, the thickness of the formations in general has been underestimated. One instance with which I am very familiar will serve to illustrate this statement. When it was proposed to drill a test-well at Ithaca, a section was published by Professor S. G. Williams which indicated that the top of the Trenton limestone would be reached at Ithaca at a depth of 2300 feet, and which stated that the bottom of the same formation would certainly be reached at a

depth of 2800 feet. The least depth at which the Trenton limestone would be reached was estimated by Professor H. S. Williams to be about 3000 feet. The Ithaca well is now 3130 feet deep and has just reached the red shale which forms the lower part of the Salina. From the records of other wells which have been drilled to the north of Ithaca, it would not seem unreasonable to conclude that the top of the Trenton limestone cannot be reached in the Ithaca well at a depth of less than 5500 feet.*

Explorations for Gas in Albany County.

The Knowersville well is located about a quarter of a mile south of Knowersville station, on the railroad of the Delaware and Hudson Canal Company, in Guilderland township, Albany county, seventeen miles from Albany. The elevation of the well is 510 feet above tide. Drilling was commenced in strata 595 feet vertically below the base of the Lower Helderberg limestone, which formation rests immediately on top of the Hudson River shales. The stratigraphical position of the top of this well is a matter of great geological significance, since it establishes the true relationship between the base of the Devonian and the top of the Lower Silurian in the Catskill region. The upper part (595 feet) of the Hudson River formation outcrops between the well and the lowest outcrop of the Lower Helderberg limestone beds near the summit of the ridge, about one mile from the well. The Upper Hudson formation constitute the base rocks of the Helderberg hills which range through Guilderland and Knox townships in Greene County.

No systematic collection of the specimens of the drilling in this well from the surface to a depth of 2500 feet was kept, but it is reported that the strata passed through were composed of gray shales and gray and black alternating slates, which in places were quite calcareous and contained occasional thin sandstones. The drilling of the well was commenced June 26th, 1886. The surface-water was cased off at a depth of 117 feet, and at 497 feet the "*gas-vein*" was struck, which is still producing gas under an undiminished confined pressure of 40 pounds per square inch. No other indication of gas was obtained in this well. A sample of this gas was collected July 19th, 1886, by Mr. J. M. Sherrerd, chemist of the Troy Steel and Iron Company. This sample was afterwards analyzed by

* Professor Prosser stands responsible for all the facts contained in his notes which end here. The arrangement and form of these notes I have materially changed in accordance with Prof. Prosser's request.

Dr. William P. Mason of Rensselaer Institute, with the following results:

Illuminants,	0.3
Carbonic acid,	0.0
Carbonic oxide,	0.0
Oxygen,	2.1
Marsh gas,	59.4
Hydrogen,	11.3
Nitrogen,	26.9
Total,	100.00

The percentage of nitrogen in the above gas is greater than is usually found in natural gas.

The gas-sand was only 6 inches thick, and at a point 50 feet above the gas-sand a pocket of gas was struck which was quickly exhausted. Great difficulty was experienced in getting sufficient water at this well for drilling purposes. In consequence a water-well was drilled to a depth of 325 feet, three feet from the deeper well. No water was got, however, in this second well, but at a depth of 200 feet a pocket of gas was struck. This gas burned to a height of 3 or 4 feet for 24 hours after having been struck, when it became exhausted. The original intention was to drill this well to a depth of 2000 feet, since it had been reported by an official of the Geological Survey of the State that the Hudson River shale formation, including both Hudson River shales and Utica slates, ranged only from 1800 to 2100 feet thick. Taking the maximum estimate as a basis for a computation, the top of the Trenton limestone should have been struck at a depth of 1500 feet.

I was not consulted in regard to this well until after the drill had gone to a depth of 2500 feet without reaching the Trenton limestone. It was then decided to drill the well to a depth of 3000 feet, and in order to ascertain the position of the Trenton limestone in the well, careful samples of all the rocks drilled through were kept, and as the drill progressed careful analyses were made of the drillings so collected. Forty-seven specimens were thus analyzed, with the results shown in the following table.

From the above analyses and also from a careful lithological inspection of the specimens themselves, it was determined that the top of the Trenton limestone was struck at a depth of 2880 feet. The drill passed through 132 feet of the Trenton limestone, the total thickness of which I have estimated to be 500 feet; but no gas was obtained in the well below a depth of 497 feet.

Samples and Analyses from the Knowersville Well.

Depth in feet from which specimens were taken.		Number of Specimens Taken.	Average Analysis.		
From	To		Calcium Carbonate.	Magnesium Carbonate.	Total.
2500	2570	6	21.16	3.69	24.85
2570	2635	4	24.34	3.49	27.83
2635	2725	4	21.79	3.93	25.72
2725	2790	2	33.21	3.11	36.32
2790	2830	3	5.96	5.23	11.19
2830	2880	4	6.52	5.23	11.75
2880	2885	1	46.96	3.36	50.32
2885	2940	5	78.75	3.36	82.11
2940	2965	4	77.50	8.69	86.19
2965	2995	6	58.39	25.49	83.88
2995	3006	3	51.96	33.45	85.41
3006	3012	2	47.76	29.96	77.72
Dark gray calcareous shale, taken at 1682 feet.....			3.63	3.82	7.45
Light gray argillaceous limestone, taken at 2230 feet.....			38.48	4.81	43.29

The geological structure of the rocks in which the Knowersville gas was obtained is such as to make it possible for them to contain gas in larger quantities than in the Knowersville well, although the fact as to whether they do or do not contain gas in greater quantity must be determined by actual drilling. The exceptional pressure of the gas which comes from the Knowersville gas-sand, which is only 6 inches thick, and the constancy in the pressure for the past 18 months, would make it probable that, if this gas-sand can be found of greater thickness elsewhere, it may contain gas in larger quantities than in this well. In order to determine whether this gas-sand was thicker elsewhere and contained commercial gas, a well was drilled on the Finch farm, in Knox township, 4½ miles north 8 degrees west from the Knowersville well.

The elevation on the top of the Knox well is 1155 feet above tide, or 645 feet above the top of the Knowersville well. The elevation of the top of the Hudson River shales and slates, at a spring about one-eighth of a mile from this well, where the Lower Helderberg limestone is seen to lie directly on top of the Hudson River shale, is 1235 feet. The average dip per mile from the Knox well to the Knowersville well is 40 feet. Between these two wells, however, there are a number of local dips, which would not materially affect the general position of the rocks, as indicated by the average dip given.

The geological horizon of the Knowersville gas-sand was passed through in the Knox well at a depth somewhere between 1000 and 1050 feet, but no gas was found. Drilling was stopped in this well at the depth of 2200 feet, or 1200 feet above the top of the Trenton limestone. Carefully selected specimens of the strata passed through in the Knox well were kept; and all the specimens from the top to the bottom of the well have such a similar character that it is impossible to define any marked change in the lithological characteristics of the rocks, which are composed of gray shales and gray and black slates, quite calcareous in places, and containing, scattered throughout the entire depth of the hole, a number of thin argillaceous sandstones. No chemical analysis was made of the specimens taken from this well and no indication of gas was obtained in it.

Geological Section of the Catskill Mountain Region.

In order to construct a complete geological column of the Palæozoic strata from the top of the Catskill mountains to the bottom of the Mohawk Valley I made numerous measurements, the details of which would be too extended to publish in this place. The following is a general section:

Carboniferous.	No. X.	Pocono or White Catskill sandstone and conglomerate, . . .		1000
Devonian.	No. IX.	Catskill red sandstone and shale, . .		2900
	No. VIII.	Chemung sandstone and shale, . .		1130
		Portage shale and sandstones, . .		1315
		Genesee shales and slates and Hamilton flags, sandstones and limestones, .		1075
		Marcellus shales and slates, . .		900
		Upper Helderberg:		
		Corniferous limestone, .	60	85
		Onondaga limestone, .	20	
		Schoharie limestone, .	5	
		Caudagalli grit, ...		40
	No. VII.	Oriskany sandstone, ...		10
	No. VI.	Lower Helderberg (and waterline), .		200
		Salina, Niagara, ...		wanting.
	No. V.	Clinton, ...		"
	No. IV.	Medina and Oneida, ...		"
	No. III.	Hudson River shales, slates and sandstones, ...		3500
	No. II.	Trenton limestone,*		500
		Total thickness of section, . . .		12,655

* One hundred and thirty-two feet of the Trenton was actually pierced by the Knowersville well. No section was measured of the strata below the Trenton.

The above section represents the aggregate thickness of the strata which might be expected to be found if a well should be drilled to a depth of 13,000 feet, commencing on the highest summits of Shandakin township, in Greene County, where some of the highest mountains of the Catskill group are contained.

The geological structure of the Hudson River Valley east of the Catskill Mountains is extremely complicated, the principal Silurian strata being thrown into numerous anticlinals and synclinals, along the slopes of which they have varying dips west of the extreme western outcrop of the Upper Helderberg limestone. Along the foot-hills the geological structure is extremely simple, consisting of monoclinal ridges and valleys in which the rocks are all found dipping to the west and northwest. The most western outcrop of the Upper Helderberg limestone has a dip in many places as high as 65 degrees. This dip declines, rapidly at first, and afterwards gradually, toward the west to a point between Lexington and Prattsville, where the rocks lie comparatively horizontal. West of Prattsville the strata dip slightly to the east and southeast.

From these dips it is evident that the rocks forming the Catskill Mountains lie in a basin with its center in a northeast and southwest line in the vicinity of Lexington and Prattsville.

From the dips to be observed along the Ulster and Delaware railroad, it would appear that this basin declines toward the southwest. They are probably only local rolls or small basins in the slopes of the main basin.

Explorations for Gas in Greene County.

The Cairo well is located in the northern part of district No 5, Cairo township, Greene County, 3½ miles in an air-line southwest of the village of Cairo, and 610 feet above tide. Black Mountain, the top of which is 3975 feet above tide, lies immediately above and to the west of the well. This well was located in the spring of 1886 by practical oil-operators from the Pennsylvania oil-regions, under the belief that the Trenton limestone would be struck at about 1150 feet, or the same depth at which it was expected to strike the top of the Trenton limestone in the well drilled at Knowersville in Albany County. As a matter of fact, however, my geological examination, made in the fall of the same year, proved that the Cairo well would have had to be drilled at least 5300 feet before the drill would have encountered the Hudson River sandstone, in which the drilling of the Knowersville well was commenced. The drilling of the Cairo

well was stopped in November, 1886, at a depth of 2200 feet. The following is a record of the well:

Well-mouth above ocean in feet, ...		=	956
1. Soil, ...	4 to	4 =	952
2. Gray flagstone, ...	3 to	7 =	949
3. Red sandstone, ...	30 to	37 =	919
4. Red and greenish gray shale, . . .	23 to	60 =	896
5. Bluish gray sandstone, ...	40 to	100 =	856
6. Green shale, ...	30 to	130 =	826
7. Red argillaceous sandstone, . . .	5 to	135 =	821
8. Gray flaggy sandstone, ...	20 to	155 =	801
9. Red sandy shale, ...	30 to	185 =	771
10. Gray sandstone, ...	15 to	200 =	756
11. Bluish gray slate, ...	25 to	225 =	731
12. Red shale, ...	30 to	255 =	701
13. Gray sandstone, ...	75 to	330 =	626
14. Red sandstone, ...	110 to	440 =	516
15. Bluish gray slate, ...	35 to	475 =	481
16. Red sandstone, ...	65 to	540 =	416
17. Gray sandstone, ...	70 to	610 =	346
18. Gray flaggy sandstone, ...	15 to	625 =	331
19. Dark gray sandstone, very arenaceous, .	125 to	750 =	206
20. Red sandstone, ...	35 to	785 =	171
21. Gray sandstone, ...	25 to	810 =	146
22. Greenish gray shale, ...	5 to	815 = +	141
23. Red shaly sandstone, ...	305 to	1120 = —	164
24. Red and gray slate, alternating, . .	10 to	1130 = —	174
25. Gray and white shale alternating, . .	90 to	1220 = —	264
26. Red shaly sandstone, ...	55 to	1275 = —	319
27. Gray slate, ...	25 to	1300 = —	344
28. Soft red sandstone, ...	150 to	1450 = —	494
29. Gray sandstone, ...	140 to	1590 = —	634
30. Very hard gray sandstone, . . .	20 to	1610 = —	654
31. Dark gray sandstone, ...	90 to	1700 = —	744
32. Hard white sandstone, ...	20 to	1720 = —	764
33. Red and gray shale alternating, . .	70 to	1790 = —	834
34. Gray shale and sandstone alternating, .	150 to	1940 = —	984
35. Gray sandstone and shale, with thin and scattered alternating beds of red and green shale, ...	260 to	2200 = —	1244

The last fresh-water vein encountered in this well was struck at 390 feet. This water was shut off by a casing set at 402 feet. A salt-water vein was struck at 610 feet. This water was subsequently shut off by a 5⅛-inch casing set at 621 feet. The amount of salt water obtained at the depth of 610 feet filled the well to a height of 300 feet in 26 hours.

The rocks cut by the drill in this well were very much harder

than those ordinarily encountered by the drill in the northern part of the Pennsylvania oil-regions, where many wells start at the base of the Olean conglomerate or the bottom member of the Millstone grit, No. XII. In a general way, the hardest strata were encountered above the point at which the salt water was found. The hardest stratum passed through in the well was the gray sandstone lying between 1590 and 1610 feet. This sandstone bed was so hard that the drillers reported it to be granite. Great difficulty was encountered in drilling the lower part of the well, which was finally abandoned, on account of the tools being lost.

It is interesting to note that the flaggy sandstone which outcrops on the turnpike, one mile southeast of Cairo, lies near the top of the Portage formation. The sandstone has a dip 3½° north, 45° west, and its surface is covered with bold glacial scratches having a direction south 10° west. This stratum was struck in the Cairo well between 1130 and 1220 feet.

What was taken to be the junction between the Portage and Hamilton formations occurs along the turnpike, where it crosses the line separating Catskill and Cairo townships. The bottom of the Hamilton formation outcrops in the vicinity of school-house No. 18, about a mile from Leeds.

The dip of the strata in the vicinity of the Cairo well precludes the existence of natural gas in commercial quantities. In the vicinity of the well the strata dip north 67° west, 185 feet to the mile. This is the minimum dip in this part of Greene County. The dip of the Corniferous limestone, where it outcrops at the town of Leeds, along the Catskill creek, is 1120 feet to the mile; the average dip from this outcrop at Leeds to the Cairo well is 580 feet to the mile.

Although gas in commercial quantities may be found in eastern New York, I have no hesitancy in asserting that it cannot be found in the region between Kingston, Catskill village and the Cairo well.

Scattered Explorations for Gas.

A number of exploration-wells have been drilled in various parts of the State. The following are scattered notes in my possession relating to some of these wells:

A natural-gas flow was struck in a well at Rodman in Jefferson County, September 4th, 1886. It is reported that this gas was found at a depth of 162 feet in a "seam of slate," limestone having been drilled through from the top of the well to a depth of 160 feet.

This limestone is evidently the upper portion of the Trenton formation.

In a well drilled at Norwich, Chenango County, which had gone to a depth of 900 feet in September, 1887, it was reported that a small show of gas was obtained in the Upper Hamilton. The elevation of the railroad station at Norwich is 1001 feet above tide.

A well is about to be drilled in Broome County, near the borders of Chenango, to a depth of 2000 feet.

A well was drilled to the depth of 1400 feet in the Neversink Valley, near the borders of Orange and Sullivan, which cost about $15,000. No indications of gas were met with and the well was abandoned.

In the town of Barker, in the Chenango Valley, a well has been drilled to a depth of 2175 feet, at a cost of $10,000. Indications are said to have been found in this well of the existence of salt, oil and gas.

At Uniontown, in Madison County, a well has been drilled which got a small amount of gas from the Lower Helderberg limestone.

At Brockport and Gasport, wells are reported to have been drilled down to the Trenton limestone.

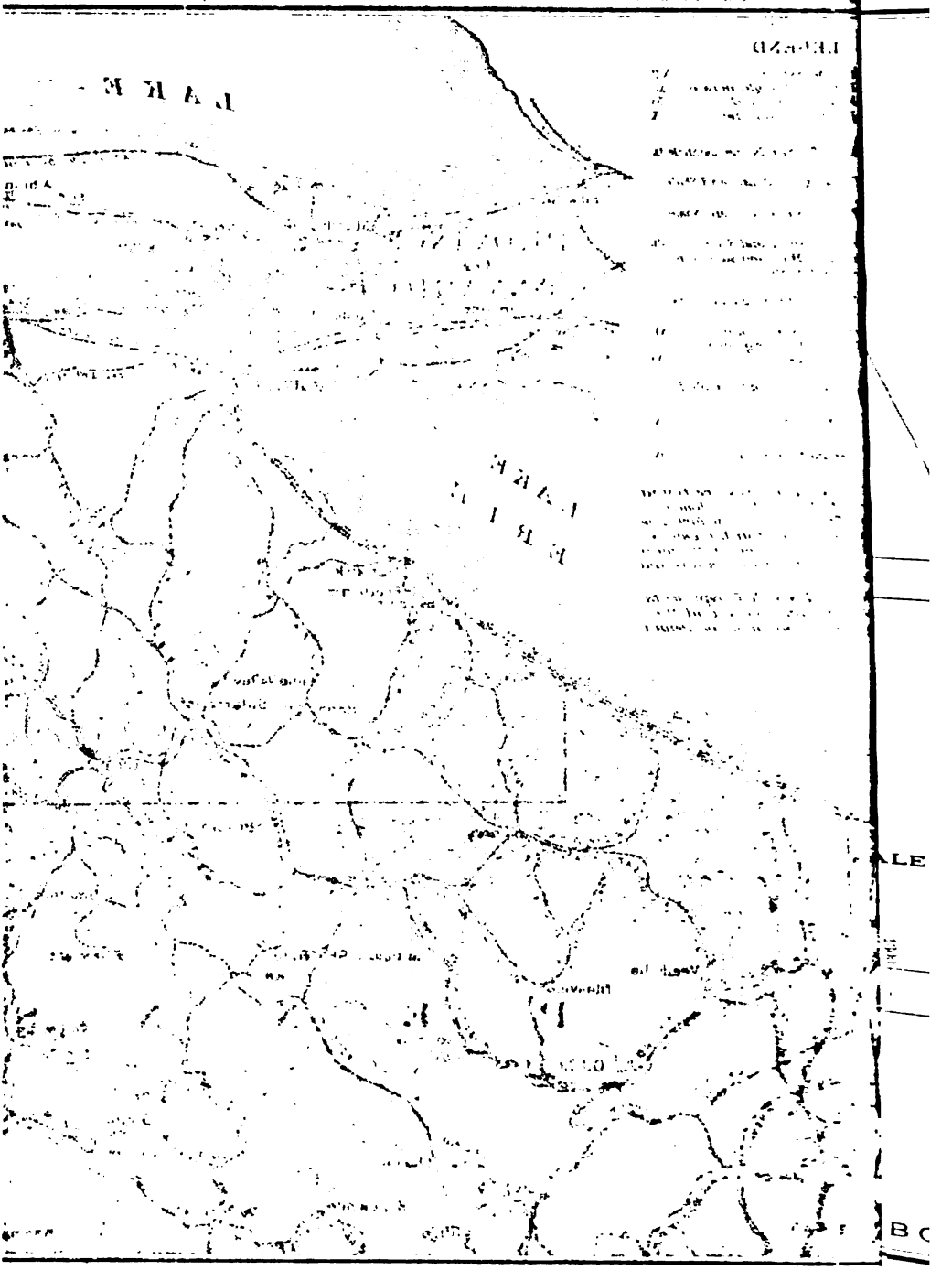
At Fulton, in Oswego County, a well has recently been drilled to a depth of 1727 feet. The elevation of the railroad station at Fulton is 387 feet above tide. This well commenced drilling in the Medina sandstone. The following is a section of the well:

IV.	Medina Sandstone,	400 feet.
III.	Hudson River shale,	880 "
	Utica shale and slate,	120 "
	Trenton limestone,	327 "
	Total,	1727 "

Mr. J. H. Case, of Fulton, reports that a small showing of gas was struck at the following depths: 42, 80, 325, 520, 585, 700, 844, 990, and 1665 feet, respectively.

At 1727 feet, 327 feet below the top of the Trenton limestone, gas was struck in such force as to throw sand from the well to the top of the derrick, a height of 74 feet 10 inches. The gas caught on fire and the derrick was burned down. This occurred during the first week of January, 1888, and for eight days after the gas was struck it burned to a height of about 2 feet.

At New York Mills, three miles west of Utica, a well has been



1300
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ALE NO. XI
BONIFI

PENNSYLVANIA RAILROAD COMPANY
AND
WESTERN NEW YORK & PENNSYLVANIA RAILWAY COMPANY
BETWEEN
JAMESTOWN, NEW YORK,
AND
BUFFALO, ROCHESTER & PITTSBURGH RAILWAY COMPANY
SHOWING THE RELATIVE LOCATIONS
OF THE
GENERAL TRACK SECTIONS
ABERN COAL MEASURES
LMEASURES - NO XIII

Transections of the American Basin

drilled, about which Professor Charles D. Wolcott reports the following facts:

"It started about 60 feet below the Oneida conglomerate and was drilled about 800 feet, passing through the Hudson and Utica shales to the Trenton limestone, and then 790 feet to what may be the Potsdam sandstone. At 2000 feet the drill entered the Gneiss. Gas was found about 500 feet down in the shales. I was not on the ground, but depended upon the owner of the well for samples of the rock and the record of the drilling." The elevation of the Erie canal at Utica is 428 feet above tide.

Mr. De Puy, of Bath, N. Y., reports the following facts:

1882. On Smith lot, half-mile south of Bath village. Depth, 2050 feet. No record kept of depth of different strata. Gas found in very small quantity—"enough to burn about two hours." Reported find of oil in small quantity, but believed to be a hoax.

1886. On James Wagner farm, half-mile west of Howard village, Steuben County. Depth, 2000 feet. No records kept as far as I can learn. No oil; no gas, except in very small quantities.

1866 or 1867. A well was sunk by oil-prospectors near Campbelltown, Steuben County. I have been able to learn nothing in regard to it except that no oil was found.

Near Cohocton, Steuben County, a well was drilled for oil, but I have not been able to see any one who could give me any definite information about it.

At Penn Yan, in Yates County, in the latter part of 1887, a well was drilled by Mr. William Townsend. Sufficient gas was obtained from this well to heat and light Mr. Townsend's house for several weeks after it was struck. It is reported, however, that the yield of the well failed a few weeks after the gas was struck.

INDEX.

ERRATA.—Special delays in obtaining the concluding papers of this volume necessitated their final publication without the usual *repeated* revision of the authors and the Secretary. As a consequence, the following corrections become necessary:

On page 799, the number of geysers in the Upper Basin should be stated, as the context shows, at 49 instead of 40.

On page 807, line 16 from the bottom, introduce "through" after "circulate."

On page 835, line 8 from the bottom, for "form" read "forming."

On page 855, in the last line of the table, the cost of wire to transmit 500 horse-power for 100 miles should be $100,000,000, instead of $40,000,000. The remainder of this table, and also the calculation below it on the same page, are affected by the correction and should be changed accordingly.

On page 912, line 2, Marysville should be Murraysville.

On page 915, line 22, minimum should be maximum.

[NOTE.—In this Index authors' names are printed in small capitals, and the titles of papers in italics. Casual references, giving little or no information about the subject, are indicated by inclosing the page-numbers in brackets. The distinction between such references and those not so marked is, however, not always sharply defined.]

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